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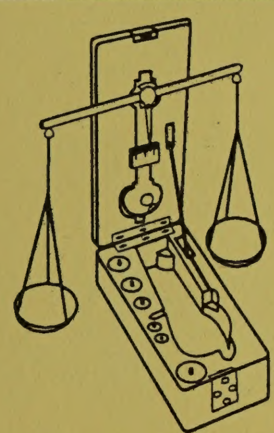
Training Center

Course Number 3000-9

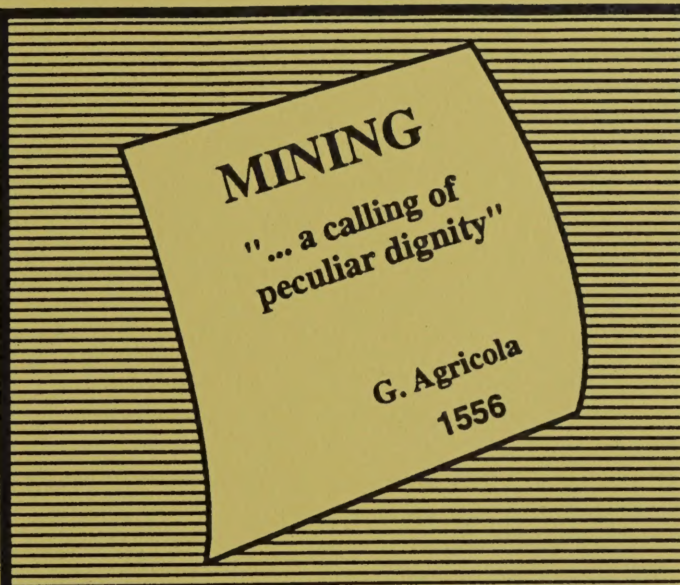
Spring, 1997

Placer Examination Techniques

Part II



A folding assay balance widely used in placer work. Capacity 10 grams, sensitive to 1/4 milligram. Complete in mahogany case with set of weights 10 grams by 1 milligram. Size when closed, about 6 x 3 x 1 1/2 inches.

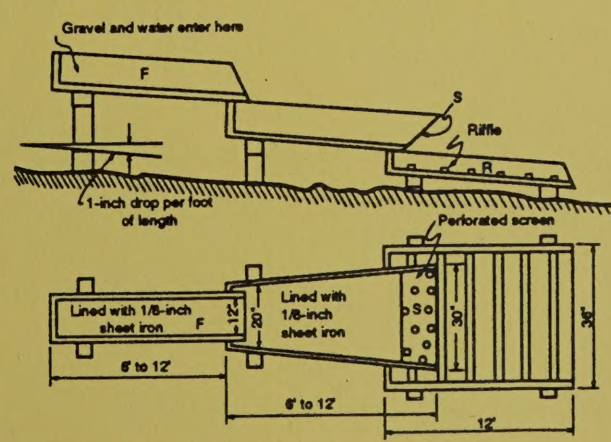


FLPMA

Section 102 (a) (12)

(a) The Congress declares that it is the policy of the United States that –

(12) the public lands be managed in a manner which recognizes the Nation's need for domestic sources of minerals, food, timber, and fiber from the public lands including implementation of the Mining and Minerals Policy Act of 1970 . . . as it pertains to the public lands . . .



Side and plan views of a long tom

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References

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CALIFORNIA STATE MINING BUREAU

FERRY BUILDING, SAN FRANCISCO

FLETCHER HAMILTON — State Mineralogist

San Francisco — December, 1918

Mines and Mineral Resources

OF

NEVADA COUNTY

BUREAU OF LA[illegible]MENT
Phoenix Training Center
5050 N. 19th Ave., Suite 300
Phoenix, Arizona 85015

By ERROL MAC BOYLE

Bureau of Land Management
National Training Center
9828 N. 31st Ave.
Phoenix, AZ 85051

CALIFORNIA STATE PRINTING OFFICE
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drifting had started from there. Another winze sunk from the 1600-foot north drift had reached 1800 feet, from which level a drift struck good ore. This winze will be continued to meet a raise from the 2700-foot level. The Champion shaft in recent years has also reached a depth of 2700 feet (inclined), but the Ural vein has evidently not come up to expectation. It is said the Champion ore-shoot was bottomed at about 1200 feet. Ore recently mined (1915) from the Ural vein has come mostly from the Nevada City ore shoot which was connected with the Champion shaft at the 1000-foot level by a drift nearly a mile long. This shoot on the latter level showed a thickness between walls of 2 to 4 feet and in places the quartz was thickly matted with sulphides of good grade. The Champion and Nevada City shafts have in the past two years become auxiliary shafts and the Providence shaft is used for hoisting ore.

The ore from this group of mines carries no specimen gold. The sulphides average 6% or more of the ore and carry 30% of the value. About 80% of the gold is saved by cyaniding, the balance by outside amalgamation. This will probably lead ultimately to the introduction of the flotation process, which has been tested here, and is thought applicable to the ore.

The Pittsburg vein, striking N. 45° E. and dipping 43° SE.; and the Gold Flat or Potosi vein, with northerly strike and dip of 40° E., are now controlled by the Pittsburg-Gold Flat Company. The shaft has been put down to a depth of 1625 feet on the Pittsburg vein. The only work of importance since the body of this report was written has been an attempt to find the Gold Flat vein from the Pittsburg shaft. On the surface the veins are about 1000 feet apart, and it was thought to be a simple matter to connect them underground by crosscutting along the line of one of the post-mineral faults. It is reported that a 1500-foot crosscut from the 1300-foot level of the Pittsburg shaft toward the Gold Flat failed to disclose the latter vein.

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Reports of the State Mineralogist, from 1886 to 1896. Folios of Geologic Atlas of the United States, U. S. Geological Survey, Nos. 18, 29 and 66. Seventeenth Annual Report, U. S. Geological Survey, part II, pages 13 to 262. The Gold Quartz Veins of Nevada City and Grass Valley Districts, by W. Lindgren. U. S. G. S. Prof. Paper 73, Tertiary Gravels of the Sierra Nevada, by W. Lindgren, 1911. U. S. G. S. Mineral Resources of the United States for the Years 1905 to the Present.

NORTH BLOOMFIELD MINING DISTRICT.

Including the region about North Bloomfield, Columbia Hill, Malakoff, Relief, Lake City, Snow Tent, Moore's Flat, Orleans, and Snow Point.

This district occurs in the central part of Nevada County, 14 miles northeast of Nevada City. The nearest shipping point is Nevada City, terminus of the Nevada County Narrow Gauge Railroad, 14 miles by a fair road which crosses the South Fork of Yuba River at a point 5 miles southwest of North Bloomfield. The elevation of North Bloomfield is 3200′.

The district occupies the divide between the Middle and South Forks of Yuba River. The climate is temperate with dry, warm summers, and heavy rains with some snow in winter. The lumber industry is important since the region is within the forest belt of the Sierra Nevada. There is an abundance of yellow and sugar pine, spruce and fir, with oak at lower elevations.

Gold is obtained extensively in Tertiary river gravels and the production is reported to have been $3,500,000 up to the year 1900.

History of mining.

Hydraulic mining has been carried on at North Bloomfield on a very large scale. Excavations from 500′ to 600′ in width, extend for 5000′ and reach a depth of 500′. The deposit has been opened by a bedrock tunnel 7874 feet long with entrance in Humbug Cañon. This tunnel together with other preliminary work is said to have cost $3,000,000. Soon after its completion hydraulic mining was hindered by anti-debris legislation and only such gravels have been worked whose tailings could be impounded before reaching the river.

The gravel produces on the average of 4¢ to 10¢ per cubic yd., the richest portions lying near bedrock. The yield between the years 1866 and 1900 was approximately $3,500,000, from the 30,000,000 cu. yds. excavated. It is estimated that 130,000,000 cu. yds. remain. A similar yardage occurred to the west, near Lake City.

At the Derbec mine, one mile due north of North Bloomfield, a shaft and workings have exposed a deep channel extending several thousand feet eastward. This channel connects with the main one of the region and has been mined for 7000 feet upstream from the shaft. The mine was operated from 1877 to 1893, and the production often reached $200,000 per year.

Hydraulic work was formerly carried on at Relief, where drift mining is now being pursued. The Union tunnel, 2500′ long, is reported to have yielded from $30,000 to $90,000 annually for a number of years.

Water is supplied by the North Bloomfield ditch, carrying 3200 miner's inches from Bowman Lake.

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Lindgren, W., Tertiary Gravels of the Sierra Nevada, U. S. Geol. Surv. Prof. Paper No. 73, pp. 139–141. Min. Res. of the U. S., 1905, p. 179; 1906, p. 191; 1907, pt. 1, p. 215. U. S. Geol. Surv. Topographic sheet of Colfax. U. S. Geol. Surv. Folio No. 66, 1900.

Topography.

The district is on a broad, comparatively flat divide between the Middle and South Forks of Yuba River. The divide is covered with andesite and characterized by elevated level areas such as Moore's Flat and Relief Hill. The descent to the rivers both north and south is steep and abrupt in places, the average slope being from 1 in 2 to 1 in 3. The region is drained by the North and South Forks and tributaries of Yuba River.

Geology.

Metamorphosed sedimentary rocks of the Delhi, Cape Horn, Relief, and Blue Cañon formations occur from west to east as broad, northwest-southeast bands. The Delhi is the more extensive formation, making up half the width of the southern end of the belt. Amphibolite extends in a north-south belt, about a mile wide through the center of the district, between the Delhi and Cape Horn formations. At the southern border of the district this belt resembles the fingers of a hand, the space between the pointed amphibolite fingers being occupied by Cape Horn slates; only the little finger, greatly elongated, extending south as a belt ¼-mile wide. Another lens-shaped body of amphibolite extends across the Relief cherts and quartzite and into the Blue Cañon formation. Small bodies of diorite intrude the Delhi formation to the northwest; one of these occurs as a long, narrow, curved band between the Delhi formation and amphibolite to the east. A small lenticular shaped intrusion of granite occurs in the Cape Horn slates about a mile east of the amphibolite belt. The Relief formation is composed of fine-grained quartzite alternating with siliceous gray slate. The general strike of the strata is from north to south, with dip nearly vertical; in detail, the stratification planes are exceedingly crumpled. Many small irregular veinlets and bunches of white quartz occur in the quartzite. The rocks of the Blue Cañon, Cape Horn and Delhi formations are further described in the reports on the Alleghany and American Hill districts, Sierra County.

The greater part of the region, especially the central part, is covered by andesite breccia, which overlies Tertiary river gravels in many places.

Diorite is a medium-grained granular rock composed chiefly of feldspar and hornblende, the amount of hornblende being equal or exceeding the feldspar. Hornblende in the diorite is of a black to dark green color, while feldspar is the light-colored constituent. Amphibolite is a schistose rock of fine-grained texture derived from dioritic rocks by pressure.

Mineral deposits.

No gold quartz veins of economic importance have thus far been reported.

Gold is found in extensive Tertiary river gravels, which are continuous with the deposits at North Columbia and Badger Hill, in the adjoining Smartsville quadrangle. Gravel is found at Lake City, North Bloomfield, Moore's Flat, Orleans, and Snow Point, possibly representing an extension of the channel from Grizzly Hill (2½ miles south of North Columbia) to Badger Hill (in the Smartsville quadrangle). A channel passing through Relief is probably the one which occurs at Omega and Alpha, in the Washington district. This joints the North Bloomfield channel, east of Columbia Hill, 1¼ miles north of the town by that name.

The bedrock in the region about North Bloomfield, consisting of the Delhi formation, rises both north and south of the main channel. The channel bed is level for nearly 400′ across; the deepest blue gravel is 130′ thick overlain by heavy-bedded, light-colored sand and clay sometimes 100′ thick; this is interstratified with fine gravel, and with andesitic tuff near the top. Unconformably above these occur 600′ of tuffaceous breccias. Most of the gold occurs in blue gravel, the richest parts being close to bedrock, but owing to the great width of the channel the gold is not concentrated sufficiently to make drift mining profitable.

The Derbec channel, one mile north of North Bloomfield, is part of that which occurs at Relief. The pay gravel at the Derbec mine was from 150′ to 600′ wide, and 8′ to 16′ deep, with an average value of $2.47 per ton. The coarse gravel contains many granite and other boulders.

At Relief this same channel reoccurs as a flat terrace on the south side of Relief Hill, and on the north side of the cañon formed by the South Fork of Yuba River. The gravels fill a deep trough in a bedrock of Cape Horn slates and cover about 200 acres. Above the terrace, andesite breccia covers the region; below, the bedrock slopes down to the Yuba River. The oldest gravels, which are coarser and contain less quartz, are 60 feet deep, and covered with from 100′ to 200′ of alternating sands, fine quartz gravel, and clay.

At Orleans and Snow Point small areas of auriferous gravel occur and hydraulicking has been carried on. The amphibolite bedrock rises rapidly to the south. Drift mining has been carried on only at Snow

Point where the gravel bank is 135′ high; the lowest fifteen feet is coarse gravel, which is in turn overlain by 20′ of clay.

At Moore's Flat, 1½ miles southwest of Orleans and 4 miles northeast of North Bloomfield two bodies of gravel are exposed. An eastern one rests on a bedrock of amphibolite, while one to the west rests on Cape Horn slates. Andesite tuff covers the gravel to the south which is similar to that at Snow Point. It varies in thickness from 100′ to 130′. Quartz boulders from 2′ to 6′ in diameter are often found on the bedrock. In 1900 it was estimated that 26,000,000 cubic yards of gravel had been washed away and that 15,000,000 cubic yards remained.

The only known quartz vein in the region is a fissure vein in amphibolite occurring one-half mile southwest of Orleans.

A mass of pyrite containing copper occurs in amphibolite on Humbug Creek, about one-half mile above North Bloomfield. Sufficient development work has not been done to determine the value of the deposit. At several places south of North Bloomfield chalcopyrite is said to occur disseminated in amphibolite.

NORTH COLUMBIA MINING DISTRICT.

The North Columbia district is known for its extensive deposits of Tertiary river gravels, and for the productive Delhi quartz vein which occurs in its northern part. It is situated near the northwestern border of Nevada County, 8 miles in a straight line and about 14 miles by a good, winding road north of Nevada City. The nearest depot on the Nevada County Narrow Gauge Railroad is Nevada City.

North Columbia lies at an elevation of 3000 ft. The summers are dry and warm, while the winters bring a heavy precipitation of rain with snow. Yellow and sugar pine, spruce, and fir are plentiful. Water is supplied by the ditch of the Eureka Lake Company and by the neighboring streams from Fancherie and French lakes.

As in the other districts in this region gold is the only mineral product of present importance.

History of mining.

At North Columbia the auriferous gravels are developed to a greater extent than at any other place. They are owned chiefly by the Eureka Lake Company, whose claims in 1900 covered an area of 1445 acres along 2½ miles of channel. Much surface work has been done and 150′ of gravel has been washed. The Delhi quartz vein was worked prior to 1893, after which it remained idle until 1898, chiefly because a great amount of water was encountered below the tunnel level. The mine was again op[illegible] up in the year 1898. The Gothardt vein has been developed by [illegible]ertical shaft 380 feet deep.

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Lindgren, W., Tertiary Gravels of the Sierra Nevada. Prof. Paper No. 73, U. S. Geol. Survey, page 139. U. S. Geol. Survey Min. Res. 1908, pt. 1, p. 341; Colfax Folio No. 66, 1900.

Topography.

The North Columbia district is continuous with the North Bloomfield district to the east, and forms part of the divide between the Middle and South Forks of Yuba River. North Columbia lies at an elevation of 3000 feet on the western side of Spring Creek. Northeast of the town the surface rises gradually to the level-topped Columbia Hill at an elevation of 4200′. The upper waters of Grizzly Creek flow parallel to the Middle Fork of Yuba River, between Columbia Hill and Grizzly Ridge. Grizzly Ridge is made up of a series of low east-west hills, about 3300′ in elevation; it constitutes a divide between Grizzly Creek and the Middle Fork of Yuba River.

Geology.

The Delhi formation, consisting of black, or dark brown siliceous metamorphosed sedimentary rocks, is intruded by diabase, diorite and gabbro, and covered by a cap of andesite breccia northeast of North Columbia. The diabase is exposed as long, slender lenses, trending in a northwesterly direction through the central portion of the district, apparently continuous with the diabase of the Pike district to the north. Diorite occurs about ½ mile east of North Columbia, also along Spring Creek to the south and near Edwards Bridge in the southwest part of the district. Gabbro occurs along Grizzly Ridge, cutting across the Delhi formation and the diabase.

The Delhi formation is the oldest in the district and was folded and disturbed along with the other sediments of Carboniferous age at the beginning and end of the Juratrias period. The igneous intrusions occurred in early Cretaceous time and the filling of the various fissures by vein material is supposed to have followed. The gravels which are now exposed in the district accumulated during the Tertiary period of erosion. At the end of Neocene, Neocene-Pliocene time, an extensive flow of andesite breccia covered the region and solidified, covering and preserving many of the auriferous gravels. During the following Pleistocene erosion period most of this andesite was worn away; considerable remains in the region northeast of North Columbia.

Rocks.

The diorite of North Columbia and Edwards Bridge is a medium-grained, granular intrusive composed of about equal amounts [illegible] light-colored soda-lime feldspar and black or dark green hornble[illegible] It is

probably part of the body of diorite exposed at the head of Grizzly Creek, north of Columbia Hill.

The gabbro on Grizzly Ridge is an extension of that in the Smartsville quadrangle to the west. It is coarse-grained, being of soda-lime feldspar, diallage, hypersthene, and olivine, and is of a dark green or black color. Near the contacts finer-grained varieties occur which in part grade into diorite.

The diabase is porphyritic, being composed of soda-lime feldspar and black or green-black hornblende or pyroxene, olivine and biotite may also occur. Where exposed it is dark green in color and of altered appearance. The fresh rock is also fine-grained and the pyroxene is often converted to uralite.

Mineral deposits.

Gold is found both as placer and lode deposits. Gravels are extensive throughout the region, but lodes occur only in one group north of Grizzly Ridge.

The placer deposits at North Columbia are the most extensive of the region. The main Tertiary river channel which branches north of North Bloomfield extends in an east and west direction through the district. The gravels are continuous with those at Badger Hill, in the adjoining Smartsville quadrangle, and form a total area covering 8 square miles. About one-half mile southeast of North Columbia a channel from the direction of Dutch Flat and Scotts Flat to the southeast joins the steeper channel from North Bloomfield; this channel was deep, with but slight grade. The North Columbia gravels are from 400′ to 500′ deep along the center of the channel, the deepest gravel being exposed at Grizzly Hill, one mile southwest of Kennebec House. The gravel there is coarse and made up largely of metamorphic rocks; the upper bench gravels being made up of finer quartzose material. Near the surface, especially near areas of andesite breccia, heavy beds of sand and light-colored clays cover the gravels. The bedrock is a black flinty rock, of the Delhi formation, and the deepest portions of the deposit can be reached only by running long and expensive bedrock tunnels. Injunctions against hydraulic mining stopped such development work. In 1900 it was estimated that 25,000,000 cubic yards of gravel had been hydraulicked away and that 165,000,000 cubic yards remained.

The veins of the region are of the fissure type; they occur in the Delhi formation, about 2¾ miles northeast of North Columbia, on the north slope of Grizzly Ridge. The principal group consists of three veins, of which the Delhi is the most important. The outcrops of the veins are parallel, having a strike nearly due north-south; that of the Delhi vein is about a half mile in length, dipping 75° E. The Delhi vein has a rich ore shoot, containing coarse gold, opened up by tunnels. Other veins in the vicinity are the Gothardt and the Live Oak. The Gothardt outcrop cut across the contact between the Delhi formation and diorite.

NORTH SAN JUAN MINING DISTRICT.

Including North San Juan, Paterson, Badger Hill, Oak Tree Ranch, Sweetland and Sebastopol.

Extensive exposures of auriferous Tertiary river gravels occurring in this district have produced a large amount of gold. An old Neocene channel extends through the district and is joined by the productive channel from North Columbia, along which large areas of gravel are exposed.

The district is located in the northwestern portion of Nevada County, with Yuba County and Sierra County bordering it on the northwest. French Corral district joins it on the south. North San Juan is 13 miles by road, in a northwesterly direction from Nevada City, the terminus of the Nevada County Narrow Gauge Railroad. Roads connect with French Corral, North Columbia and Camptonville. The greater part of the region is between 2000′ and 3000′ in elevation. Vegetation consists of yellow pine and various kinds of fir and spruce. The principal source of timber is in tracts of the Oregon Hills. North San Juan is furnished with water by the Eureka Lake Company's ditch, carrying water from the headwaters of Middle and South Yuba rivers. Shady Creek flows from Sugar Loaf Hill through the district.

History of mining.

It has been estimated that 20,000,000 cubic yards of gravel had been excavated before the year 1891, leaving 2,500,000 cubic yards still available. Almost continuous deposits extending from North San Juan to French Corral have been worked throughout their extent, and large portions have been exhausted.

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Topography.

The topography of the district consists of low hills, between the Middle and South Forks of Yuba River, rising from the general plateau, the elevation of which varies from 2000′ to 2500′ above sea

level. The area rises eastward to Sugar Loaf Hill which reaches an elevation of 3300′.

North San Juan is on the plateau in the northwestern part of the district, near the steep south side of the cañon of the Middle Yuba River. The cañon near North San Juan is 500 feet deep, while at Badger Hill (elevation of 2500′) on the edge of the cañon it is 900′ deep.

The southern part of the region is drained in a southwesterly direction. The waters of Shady Creek flow from the region about Sugar Loaf draining Paterson and Oak Tree Ranch, and emptying into the South Fork of Yuba River at a point southeast of French Corral.

Geology.

The greater portion of the region is composed of granodiorite which covers an area over 4 miles wide in the North San Juan district. The belt is part of that which extends northwestward from the Nevada City region. It is a light-colored rock of uniform texture and consists of quartz, large grains of black hornblende, black mica, much plagioclase and small amounts of orthoclase. Between North San Juan and Freeman's Bridge it carries some muscovite.

The eastern contact of granodiorite with the Calaveras formation enters the southeastern corner of the district and can be followed in a northwest direction towards Paterson to a point 2¼ miles northeast of North San Juan; thence east for 1¾ miles and then northward. North and southeast of Paterson the Calaveras formation, in places, is in contact with diorite. The contact of this diorite with the granodiorite to the west is not sharply defined.

The western contact of granodiorite with amphibolite schist has a north-south direction west of Sweetland. The schistosity of the amphibolite strikes northwest and dips 68° E. At a considerable distance westward from the contact the amphibolite changes to augite-porphyrite.

The Calaveras formation is composed of closely folded clay slates and quartzites which have been crystallized along the contacts to micaceous and quartzose schists by igneous intrusions. This altered zone is rarely over half a mile in width and there is a gradual change to unaltered rock. Northeast of Badger Hill the Calaveras formation strikes northwest and dips 72° NE. It is a part of a continuous area of sedimentary rocks of the middle slope of the Sierra occurring much more extensively in the Colfax quadrangle to the east. It is the oldest formation in the district, being of Carboniferous age, and was probably folded and compressed at the end of the Juratrias period when the first great uplift of the Sierra Nevada mountains took place. The grano-

diorite and gabbro-diorite were intruded about this time, the granodiorite being somewhat younger.

The relative ages of the granodiorite, the augite-porphyrite and the amphibolite schist have not been definitely determined, but the granodiorite of Nevada City, which is of the same period as that in the North San Juan district is known to be later than the diabase rocks to the south and west of it. According to the petrographic character of a large portion of the augite-porphyrite in the Smartsville quadrangle it should be considered as a heavy flow. The amphibolite schist was probably formed by dynamic-metamorphism of the augite-porphyrite. The dynamic action was intensified along two shear zones, one of which passes through Birchville.

Auriferous gravels accumulated during Neocene erosion. At the end of Neocene time flows of andesite breccia occurred, since which the region has been extensively eroded. Auriferous gravels, possibly derived from the Tertiary gravels to the east, accumulated near the head of Shady Creek, south of Paterson, during late Pleistocene time.

Mineral deposits.

Gold is the chief mineral deposit, occurring extensively in the Tertiary river gravels. The Neocene Channel of Yuba River possibly extended from Camptonville to North San Juan, and then southwest to French Corral. About a mile north of North San Juan the North Columbia channel, turning through Paterson, joined the main channel. No gravels are found at the juncture, since they have been washed away by the more recent drainage system.

The gravel deposits at Paterson and North San Juan are composed of well-rounded pebbles of quartz, siliceous metamorphic rocks and some sand. The gravel beds between North San Juan and French Corral average 150′ thick, while east of Paterson they reach a thickness of 400′. Extensive hydraulic mining has been carried on both here and at Badger Hill, but drifting operations had not been undertaken because the gravel was considered of too low grade. A bedrock, of Calaveras formation, was exposed in the center of the channel at Badger Hill in 1895. The gravel deposit west of North San Juan is about a mile in length and rests on granodiorite. The deposit south of this is longer and of greater width; the bedrock changes from granodiorite to amphibolite schist. None of the gravel deposits in the district are covered with andesite breccia. The grade of the channel from North San Juan to French Corral is 65′ per mile; the Badger Hill channel is almost level. Three miles north-northwest of Montezuma Hill, on the hills west of the Oak Tree Ranch, is a deposit of well-washed gravel resting on granodiorite bedrock at a higher level than

the gravel channel of North San Juan. This deposit is probably of an earlier period than the Tertiary gravels.

In the belt of amphibolite schist extending from Birchville north to Bullards Bar several quartz veins have been found carrying auriferous iron and copper pyrites. One of the larger veins strikes north and dips 80° E.

ROUGH AND READY MINING DISTRICT.

Including the region about Rough and Ready, Anthony House, Rapps Ranch, and Newton.

This district is adjacent to the famous Grass Valley district, and produced a considerable amount of gold, chiefly from gravels deposited in a branch of the Neocene channel of Yuba River. The district includes a group of workable auriferous quartz veins.

Rough and Ready is situated in the west central portion of Nevada County, 3½ miles by road west of Grass Valley, a point on the Nevada County Narrow Gauge Railroad, 12½ miles from Colfax. The town lies at an elevation of 1800 feet above sea level. The climate is mild, with moderate rainfall and some snow during winter time. The foothill region to an elevation of 2000′ is generally covered by digger pine and oaks. Above 2000′ yellow pine, spruce and fir predominate. The principal timber occurs as a tract in the Oregon Hills about 20 miles to the north.

There is a good water supply. Squirrel Creek, Deer Creek, and Clear Creek flow west to Yuba River. Ditches of the South Yuba Canal Company carry water from the headwaters of South Yuba River to Grass Valley for mining and other purposes. Electricity may be used for power.

History of mining.

In the year 1891 it was reported that 3,000,000 cubic yards of gravel had been excavated at Rough and Ready and Randolph Flat, and that 1,000,000 cubic yards remained available. In 1909 the total production of the hydraulic mines at Rough and Ready, French Corral, Smartsville and Grass Valley was only $8,000.

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Topography.

The region in general is broken by groups of hills, none higher than 2500′ above sea level, which separate a series of parallel westward flowing streams. Squirrel Creek flows northwest through Penn Valley to a point a mile west of Anthony House, where it joins Deer Creek. Northwest of Penn Valley the ground slowly rises to hills 500′ to 600′ above the valley floor. Rough and Ready lies near a tributary 2 miles northeast of Squirrel Creek and 300 feet above Penn Valley. Groups of hills constitute low parallel ridges, ascending eastward to the Grass Valley plateau, about 2500′ above sea level. The north side of the Rough and Ready divide is drained by Deer Creek, north of which the ground rises to a series of parallel hilltops, the highest of which is about 700 feet above the creek.

In the southwest portion of the district, south of Penn Valley, Indian Springs plateau extends eastward from Indian Springs Hill. The elevation of this plateau averages 1800′. It is drained on the north by Clear Creek, which flows northwest to join Squirrel Creek. East of the plateau the surface is made irregular by a number of low hills.

Geology.

There is one small area of slates and quartzites of the Calaveras formation in the northeast corner of the district, being continuous with those formations of the Grass Valley region to the south. The region is composed chiefly of igneous rock. The three most extensive formations are granodiorite, gabbro-diorite, and amphibolite, occurring from west to east in the order mentioned. To the west the granodiorite is in contact with diabase; it occurs in and north of Penn Valley about Anthony House and at Rapps Ranch. It also occurs north of Rough and Ready, in Deer Creek Cañon, Kentucky ravine and on the intervening divide. Somewhat over a mile east of Rapps Ranch a body of gabbro-diorite, one mile in greatest width and four miles long, grades into the surrounding granodiorite. Southwest of Rough and Ready is a large body of gabbro-diorite, two miles in greatest width, in contact on the west with granodiorite and on the east with amphibolite. Gabbro-diorite forms the southern end of a large area of granodiorite enclosed in the main diabase area in the northwestern part of the Smartsville quadrangle. Frequent transitions from granodiorite into adjoining gabbro-diorite are found, showing that the latter is probably a magmatic segregation.

Amphibolite is in contact with gabbro-diorite both east and south of Rough and Ready. It is massive and is thought to have been derived from gabbro-diorite. The texture is not the same throughout the area, and masses of diorite and gabbro are often included. The change to amphibolite is brought about by pressure, which changes the pyroxene to green uralitic hornblende. When the change is complete the amphibolite is composed of secondary amphibol, albite, epidote, chlorite and other minerals. The amphibolite belt has a width ranging from 2 miles

through the shaft and rich gravel was taken from a paystreak 50′ in width. The Ragon incline, through which the work at present is being carried on, lies about 500′ east of the Empire shaft. The incline was put down at an angle of 45° north and is 145′ in depth. From the bottom a drift has been driven 130′ northwest and 60′ southeast, and short branch drifts have been run across the channel from the main drift.

The width of the pay gravel has not as yet been determined, but some of the gravel encountered is said by Mr. Graham to average from $3 to $4 per cu. yd. and the depth to be 4 feet. The gravel is in part cemented, and it is the intention of the company to install a ten-stamp mill. The property at present is equipped with a 22 horsepower electric hoist and a 6″ electric driven Cornish pump.

This is the only company operating on this channel at the present time.

Manzanita Mine. Owner, Fred Ayer; W. F. Duboise, agent, Nevada City.

Location: Nevada City Mining District, Secs. 6 and 7, T. 16 N., R. 9 E., 1 mile north of Nevada City. Elevation 2800′.
Bibliography: Cal. State Min. Bur. Repts. VIII, page 458; XI, page 302; XII, page 191; XIII, page 251. U. S. Geol. Survey Folios 18 and 28. U. S. Geol. Survey Prof. Paper 73.

The Manzanita channel was worked in early days leaving the hydraulic pit as one of the prominent features of topography north of Nevada City. After the suspension of hydraulic mining, the rich gravel lying on the bedrock of the channel was worked by drifting from Nevada City under the lava-capped ridge to the northern exposure of the channel in Howe cut, a distance of 3000 feet. The central portion of the channel under the deep lava cap was worked by means of the Odin and Nebraska inclines in the Odin property, which adjoins the Manzanita on the north. This channel is reported to have produced over $3,000,000. The Harmony channel, which has been extensively worked in the East and West Harmony mines, about a mile northeast of Nevada City, was formerly supposed to be connected with the Manzanita channel under the lava-covered Harmony Ridge. Now the data obtainable seems to point to the conclusion that the Manzanita and Harmony channels both flowed in a northerly direction and their point of juncture was one or two miles north of Howe cut. This portion of the ancient streams has been eroded.

Murphy. (See Grover.)

Sazarack Claims. Owner, E. A. Roberts Estate; W. H. Lyons, agent, Stockton, California.

Location: Rough and Ready Mining District, Sec. 19, T. 16 N., R. 8 E., 3 miles west of Grass Valley. Elevation 2000′.
Bibliogr[illegible] U. S. Geol. Survey Folio 18, Smartsville.

This property consists of the Sazarack and Bunker Hill patented placer claims, an area of about 120 acres. These claims are supposed to contain the western extension or a branch of the Alta Hill-Town Talk Channel. The bedrock is amphibolite schist, in part covered with a volcanic capping of andesite. No work has been done on the claims for a number of years.

Sharpe Quartz and Gravel Mining Company (Black Bear). (See under Lode.)

Union Mine. Owners, C. D. Jepsen et al., Relief Hill.

Location: North Bloomfield Mining District, Secs. 4 and 9, T. 17 N., R. 10 E., 3½ miles east of North Bloomfield, thence 14 miles by road southwest to Nevada City. Elevation 3600′.
Bibliography: Cal. State Min. Bur. Repts. XII, page 201; XIII, page 265. U. S. Geol. Survey, W. Lindgren, Prof. Paper 73, pages 139–141. U. S. Geol. Survey Folio 66, Colfax.

The Union mine covers gravels of the Dubec channel. At Relief erosion has exposed a deep trough in the old bedrock and about 200 acres of auriferous gravels. The oldest gravel, as usual coarser and containing less quartz, is 60′ deep and covered by 100′ to 200′ of alternating quartz sand and clay.

The Union tunnel, about 2500′ long, has been driven from the southwest side of the gravel area, and amounts up to $30,000 and $40,000 were produced for a number of years.

In December, 1915, the tunnel was being driven to intersect the bottom of an old 40′ incline shaft that was sunk to the gravel channel.

West Harmony Claim. (See Delaware.) Owners, C. H. Mallen et al., Nevada City.

Location: Nevada City District, Sec. 5, T. 16 N., and Sec. 32, T. 17 N., R. 9 E., 2 miles northeast of Nevada City. Elevation 3000′.
Bibliography: Cal. State Min. Bur. Repts. XI, page 300; XII, page 202; XIII, page 210. U. S. Geol. Survey Folio 29, Banner Hill Special.

Yosemite. (See Albert.)

You Bet Mining Company. Owner, Birdseye Creek Mining Company, Nevada City; Geo. Wight, Nevada City, president; C. P. Banning, secretary.

Location: You Bet Mining District, Secs. 6, 26, 28, 30, 31, 32, 36, 25, and 36, Twps. 15 and 16 N., R. 10 E., 7 miles northwest of Dutch Flat. Elevation 2600–3200′.

(See under Hydraulic Mining for activity planned in 1918.)

GOLD—HYDRAULIC MINES.

At North Bloomfield and North Columbia hydraulic mining was developed and prosecuted on a very large scale from 1865 to 1880. The immense amount of gravel discharged into the Yuba River from the hydraulic mines operating from North Bloomfield to French Corral was

the main cause of the agitation against hydraulic mining by the inhabitants of the Sacramento Valley, which culminated in the suspension of this method of mining in 1894.

The largest operations were carried on at North Bloomfield where from 1866 until the closure of the mine, February 1, 1894, the production had been \$2,829,869. From 1894 until 1900 the property was worked at intervals and the total production to date is said to have been about \$3,500,000 from 30,000,000 cubic yards of gravel. The channel of the Neocene river at this point was from 500′ to 600′ in width. The bank washed reached a maximum height of 500 feet, but most of the gold was obtained from the first 150′ above bedrock. This gravel averaged over 10¢ per cubic yard.

At North Bloomfield it was necessary in order to secure sufficient grade, to drive a tunnel from Humbug Creek 8875 feet in length. This tunnel was completed only a short time before the adverse decision in the suit of *Woodruff* vs. *The North Bloomfield Gravel Mining Company*, practically stopped hydraulic mining in the Sierra Nevada. Following the formation of the California Debris Commission by act of Congress in March, 1893, hydraulic mining was resumed on a small scale at North Bloomfield. The increasing severity of the restrictions passed by the commission finally resulted in a complete cessation of work in 1900.

From 1866 to 1900 about 30,000,000 cubic yards had been worked in the vicinity of North Bloomfield and the total production therefrom was approximately \$3,500,000. It is estimated that 300,000,000 cubic yards, which will average over 10¢ per yard, still remain available for hydraulic mining in the North Bloomfield-Lake City area.

Near North Columbia the auriferous gravel deposits are extensive, owing to the junction of two of the large Tertiary streams near this point. The gold-bearing gravel at this point is from 400 to 500 feet in depth. It is estimated that 165,000,000 cubic yards still remain to be worked on the holdings of the Eureka Lake Company, which cover an area of 1445 acres and a distance of 2½ miles along the channel. In the former operations only 25,000,000 cubic yards of the top gravel had been removed. It was planned to work the lower and richer gravels by means of a long bedrock tunnel, but on account of the suspension of hydraulic mining this was never done. The remaining gravel is said to run from 10¢ to 25¢ per cubic yard.

West of North Columbia the channel of the Tertiary Yuba River turned abruptly to the North. Beyond Badger Hill it has been eroded by the present Middle Yuba River, which has cut down into the granodiorite to a depth of 1000 feet below the bottom of the old Neocene stream.

From North San Juan to French Corral, the ancient river was preserved and the gravel deposits of this area have been extensively worked. The channel here reaches 1000 feet in width and the gravel averages 150 feet in depth. The top gravels here yielded from 10¢ to 15¢ per cubic yard, while the average value is said to have been 30¢. It is estimated that 52,500,000 cubic yards have been washed and that 12,500,000 cubic yards are still available. The production of the property to 1884 was \$3,412,509.

From 1870 to 1882 over \$4,000,000 was expended by the North Bloomfield Gravel Mining Company, its subsidiary companies, and the Milton and Union Mining Companies, in opening up the deposit, construction of reservoirs and ditches, and finally in driving bedrock tunnels.

The Bowman's Lake and other reservoirs, together with the 100 miles of distributing ditches, which were constructed at a cost of from \$8000 to \$10,000 per mile, are now owned by the River Mines-Eureka Lake companies.

It has been estimated that between \$75,000,000 and \$100,000,000 in gold lies buried in the gravels of the Neocene streams between North Bloomfield and French Corral.

This gold could probably be recovered without damage to the navigable streams or agricultural districts of the lower valleys, by using modern engineering methods in the construction of restraining dams in the lower reaches of the Yuba River at different points where natural dam sites exist. The first cost of such restraining dams would be large and it seems only reasonable that financial aid be given by state and federal governments for their construction. Such works would assist materially in solving many of the problems now being faced by the agricultural, irrigation, and navigation interests.

Badger Hill. (See River Mines Company.)

Baltic. (See under Lode.)

Beckman. (See Mayflower, Lode.)

Birdseye Creek. (See You Bet Mining Company.)

I. X. L. Claim. Owners, James Howlett, North Columbia; H. C. Dillon Estate, 2850 Leeward Avenue, Los Angeles.

Location: North Bloomfield Mining District, Secs. 3 and 10, T. 17 N., R. 9 E., 3 miles west of North Bloomfield.
Bibliography: U. S. Geol. Survey Folio 66, Colfax. U. S. Geol. Survey Prof. Paper 73, W. Lindgren, Tertiary Gravels of the Sierra Nevada.

The I. X. L. placer claim is an elongated area of about 100 acres situated a mile northwest of Lake City. The famous North Bloomfield tertiary channel is in part covered by this claim, but owing to the suspension of hydraulic mining no work has been done for many years. For description of the channel see River Mines Company.

Liberty Hill Mines. Owners, Wm. Maguire and Phœbe Maguire (one-half), and Anna F. Smith (one-half); lessees (with option to purchase) W. S. Bliss et al., 611 Insurance Exchange Building, San Francisco. The properties consist of the Liberty Hill Placer Mine (patented), Lots 41 and 66 in Sec. 23, T. 16 N., R. 10 E., 494.97 acres; the Maguire placer locations, containing about 160 acres in the same section; the Little York Placer Mine (patented), which includes Lots 37 and 38 in Sec. 5, T. 15 N., R. 10 E.; and Lots 39, 40 and 76 in Sec. 33, T. 16 N., R. 10 E., 430.9 acres; an undivided one-half interest in

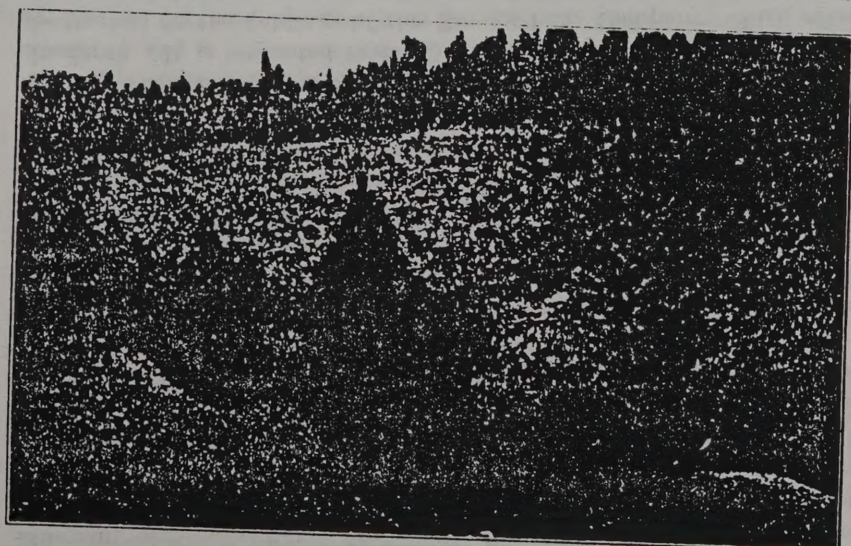

Photo No. 10. Liberty Hill Hydraulic Mine. Photo by C. A. Logan.

the Consolidated Junction Placer claim in Secs. 26, 27, 34, 35, T. 16 N., R. 10 E., in Placer County, 75.41 acres; and an undivided one-half interest in the Liberty Hill and Polar Star Tailings Dam in Bear River, half a mile from Dutch Flat, being in all 1161.28 acres.

Location: The Liberty Hill property is about 3 miles by trail or 7 miles by road from Dutch Flat on the opposite side of Bear River, at an elevation of 3349 feet. The Little York Mine is about 4 miles southwest of Dutch Flat by road.

Lindgren* regarded the Little York deposit as pretty well worked out, but estimated that of a total of 18,000,000 cubic yards at Liberty Hill only 1/9 had been hydraulicked.

The channel is said to be 600 yards wide at Liberty Hill. The gravel is 60′ to 85′ deep and without overburden. It contains some very heavy boulders of gabbro and related rocks and is generally loose. The

*Lindgren, W., Tertiary Gravels of the Sierra Nevada of California: Prof. Paper 73, U. S. Geol. S[illegible] pp. 144–146.

bedrock in this section of the old stream, rises rapidly. The property is equipped with 45 miles of ditch, the longest being 9 miles long and delivering the water under 250 feet head. The ditches are largely intact, but will require considerable cleaning, as will also the reservoir. The main pipe line into the workings is apparently in good condition. The property has first rights to water out of Bear River, and the normal supply will permit six months mining.

Photo No. 11. Log crib, hydraulic fill dam, formerly used to restrain tailings from Liberty Hill and Polar Star hydraulic min[illegible] being raised in November, 1918, preparatory to renewed activ[illegible] at former mine. Located in Bear River one-half mile from Du[illegible] Flat. Photo by C. A. Logan.

A force of about twenty men were at work in September, 1918, raising the dam on Bear River, preparatory to hydraulicking the Liberty

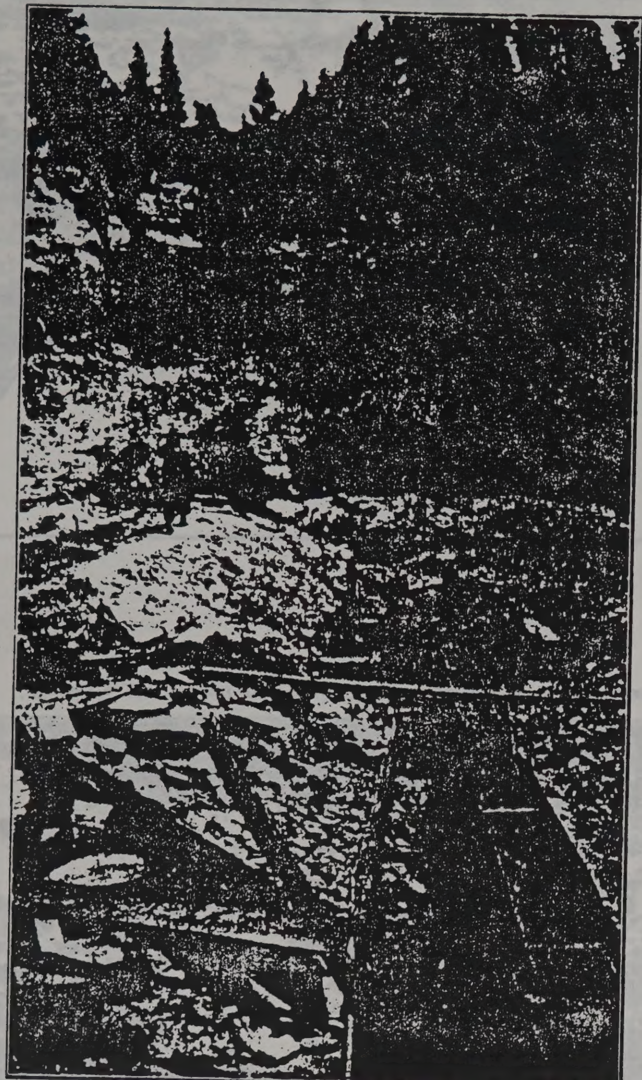

Photo No. 13 A. Chinamen cleaning bedrock at the Omega Hydraulic Mine.

Hill property the following winter. Hydraulic mining with one giant and about 1200 inches of water began January 15, 1919.

Manzanita. (See under Drift.)

Mayflower. (See under Lode.)

Photo No. 12. General view of the Omega Hydraulic Mine, near Washington, looking north. Shows particularly the manner of piping the water to the working banks.

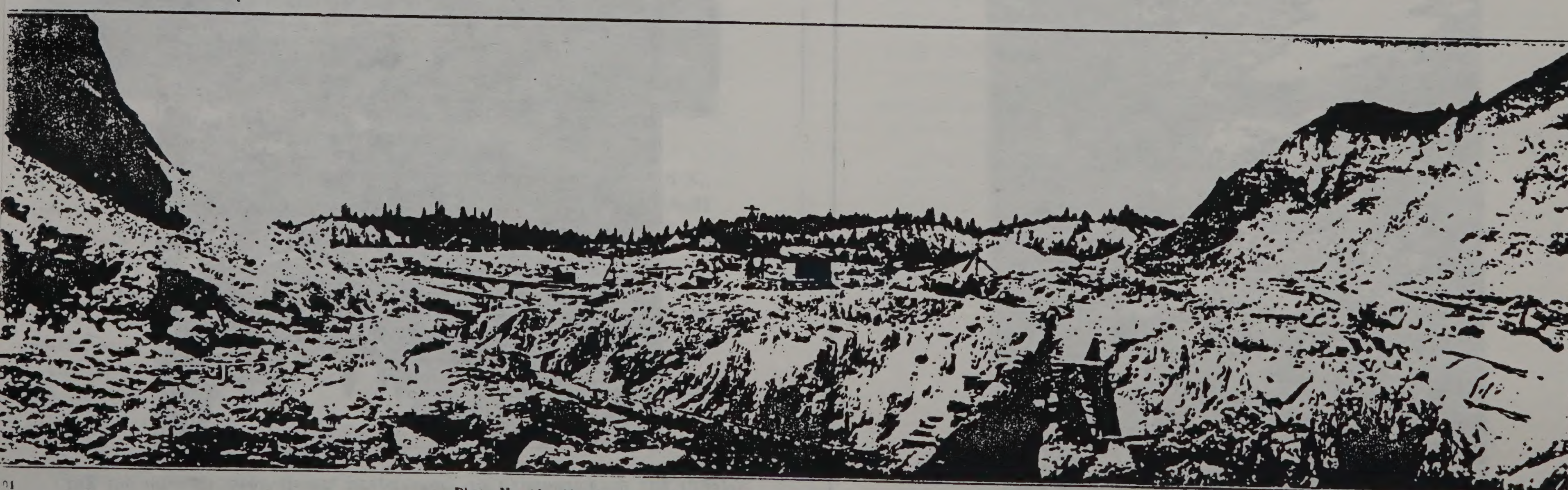

Photo No. 13. Nearer view of Omega Hydraulic Mine showing extent of gravel remo.ed.

North Bloomfield. (See River Mines Company.)

Odin. (See Manzanita, Drift.)

Omega Mine (Prescott). Owner, Omega Placer Mining Company, 1213 Fischer Building, Chicago, Illinois.

Location: Washington Mining District, Secs. 16 and 17, T. 17 N., R. 11 E., 3 miles southeast of Washington. Elevation 4000'.
Bibliography: Cal. State Min. Bur. Rept. XIII, page 258. U. S. Geol. Survey Folio 66. U. S. Geol. Survey Prof. Paper 73, page 147.

The Omega hydraulic mine was worked for many years before and after the restriction of hydraulic mining. The accompanying photograph will give a good idea of the amount of material which has been

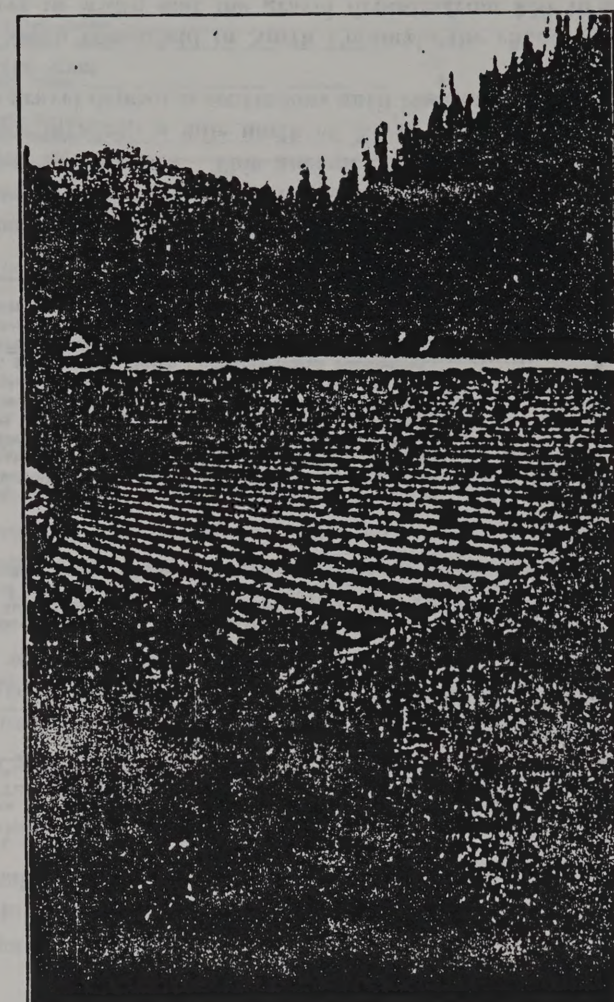

Photo No. 14. Debris restraining dam of the Omega Hydraulic Mine near Washington. Photo by C. A. Logan.

worked. About 13,000,000 cubic yards have been removed which is said to have averaged 13½ cents per cubic yard, and it is estimated that 40,000,000 cubic yards remain which can be worked. In late years, up to about 1915, most of the operations have been carried on by Chinamen under a leasing system.

The gravel deposit is a portion of an old Neocene river and the ground is 175 feet in depth. The elevation of the bedrock is 4028′ or 1000 feet above the bed of the South Yuba River. There are two strata of gravel. The bottom 150 feet is small gravel with a large amount of quartz, the greater part of which does not exceed 6 inches in size, although some large boulders of granite are included. Above this there was from 6 to 10 feet of fine pipe clay, overlaid by another 20 feet of fine gravel, which extends in a southeasterly direction under a capping composed of tuffs and volcanic breccia of andesite. The bedrock is Calaveras slate. Twenty Chinamen were at work cleaning up bedrock when the mine was visited in 1914 but no information could be obtained from them regarding production, costs, etc. A new restraining dam was being constructed below the 3000-foot drain tunnel.

Since 1915 the property has been worked by two partners during the water season. The photo of the restraining dam shows that it is filled nearly to capacity and must be raised if more mining is done. Raising the dam and mining in 1918 is planned.

River Mines Company (Eureka Lake and Yuba Canal Company). Owner, River Mines Company, care Geo. W. Starr, Grass Valley, California.

Location: Claims in French Corral, North San Juan, North Columbia and North Bloomfield Mining districts. Elevation from 2000′ to 4000′.
Bibliography: Cal. State Min. Bur. Repts. U. S. Geol. Survey Folios 18 (Smartsville) and 66 (Colfax). U. S. Geol. Survey Prof. Paper 73, Tertiary Gravels of the Sierra Nevada, Lindgren.

A partial list of the claims now controlled by the River Mines and Eureka Lake Companies is as follows:

Name	Location, mining district	Sections	Township	Range
River Mines Company	French Corral	12, 14, 24, 25, 35	17 N.	7 E.
Bed Rock Tunnel Company	French Corral	14 and 23	17 N.	7 E.
Kate Hayes Placer Mine	French Corral	25 and 26	17 N.	7 E.

	Section	Township and range
Milton, placer mine, French Corral	26	17N., 7E.
Badger Hill and Cherokee, placer mines, North Columbia	1	17N., 8E.
	36	18N., 8E.
American, placer mine, North San Juan	6, 7	17N., 8E.
Sebastopol, Sweetland, Bloomfield Hydraulic Gravel, placer mines, North Bloomfield	1, 2, 11, 12	17N., 9E.
North Bloomfield Gravel Mining Co., North Bloomfield	1	17N., 9E.
	35, 36	18N., 9E.
	6	17N., 10E.
	31	18N., 10E.
Consolidated, placer mine, North Columbia-North Bloomfield	5, 7, 8	17N., 9E.
Central Gravel, placer mine, North Columbia-North Bloomfield	5, 7, 8	17N., 9E.
Northern Gravel, placer mine, North Columbia-North Bloomfield	5	17N., 9E.
Western Gravel, placer mine, North Columbia-North Bloomfield	6, 7	17N., 9E.
Union Gravel, placer mine, North Columbia-North Bloomfield	9, 10	17N., 9E.
Humbug Creek, placer mine, North Columbia-North Bloomfield	12, 12, 14	17N., 9E.
Relief Hill, placer mine, North Columbia-North Bloomfield	4, 9	17N., 10E.
Waukashaw, placer mine, North Columbia-North Bloomfield	6	17N., 10E.
Cooke & Porter, placer mine, North Columbia-North Bloomfield	6	17N., 10E.
Snow, placer mine, North Columbia-North Bloomfield	13, 14	18N., 10E.
Hazard, placer mine, North Columbia-North Bloomfield	16, 21	18N., 10E.

The above claims control the major portion of the Great Neocene Yuba River from North Bloomfield to French Corral, a distance of approximately 15 miles. This ancient stream emerges from beneath an andesite lava cap a mile north of North Bloomfield and from this point the gravel deposit is continuous until Badger Hill is reached, eight miles to the west.

From North Bloomfield to North Columbia the channel is from 300 to 600 feet in width and the gravel deposit from 150 to 500 feet in depth. Portions of the deposit are covered by sand and light-colored clays. Above North Bloomfield the main channel branches; one branch continues eastward toward Relief Hill and the other northward toward the Middle Yuba River. These channels lie under a capping of lava and the easterly or Derbec channel has been worked by drifting for a distance of 7000 feet up stream.

You Bet Mines. Owner, You Bet Mining Company; lessee (with option to purchase), California Placer Mining Company, W. L. McGuire, secretary, tenth floor Crocker Building, San Francisco.

The You Bet Mining Company started hydraulic mining in the winter of 1913–1914, after building a debris dam and getting a permit from

the Debris Commission. The mine was worked successfully till August, 1914, when work was stopped by a Court injunction, it being claimed that the water, which was used below the dam for domestic purposes, was being rendered unfit for use because of its turbidity. In August, 1915, many of the claims were leased to Chinese, who began drift mining near the old 'tie' workings near You Bet.

The present lessees took the property in the summer of 1918. The dam in Missouri Cañon is to be raised 15 feet. It is a gravel dam with concrete spillway. The property has 15 miles of ditches and a maximum water supply of 1200 inches. There is said to be about ¾-mile of unworked channel between the old Hayward shaft and the Nevada tunnel. There are three pipe lines into the workings under heads of 100, 200, and 300 feet respectively.

The work will be done mostly in cemented gravel, which is from 60 feet to 100 feet thick. Powder drifts and crosscuts will be driven, loaded with powder and shot to break up the cement. Probably three giants with 5″ and 6″ nozzles will be used in piping and a 5″ giant will be used to stack tailings. The property is equipped with 900 feet of 6-feet sluice split into two 5-feet sluices at the pit and with iron rails and block riffles. Twenty men were employed in September, 1918, preparing for mining. It is said that this mine has yielded 40¢ a yard in hydraulic mining and $7 to $9 a car in drift mining. It is estimated that 50,000,000 cubic yards of gravel have been worked in the past and that 100,000,000 cubic yards are available. The You Bet hydraulic mines are said to have produced about $3,000,000, in addition to yield since 1913, for which figures are not obtainable. It is known that operations in 1913–1914 paid well, however.

The gravel of the deep channel in the vicinity of You Bet is coarse and cemented in the deep trough, but in the upper portions is fine quartz gravel mixed with sand. In the deep channel the gravel is from 75 feet to 200 feet deep, and on the benches from 90 feet to 100 feet, and the width of pay gravel is 300 to 400 feet.

The following is a list of the claims held by this company:

	Section	Township	Range	Area
Birdseye Canyon placer mine	6	15	10	
	31	16	10	19.65
Walloupa Canyon placer mine	6	15	10	
Arkansas placer mine	26	16	10	
	25	16	9	35.60
Greenhorn placer mine	30	16	10	
	6	15	10	
Washington placer mine	31	16	10	37.94
Red Dog placer mine	30	16	10	48.03
You Bet placer mine	31	16	10	107.01
Neece & West, Brown's Hill	6	15	10	
Walloupa	31	16	10	87.59
Starr	25	16	9	21.84
	30	16	10	
Mallory and other claims	-----	-----	-----	5.37
Brown Bros. placer mine	6	15	10	28.65
	31	16	10	
Poverty placer mine	6	15	10	60.86
	31, 32	16	10	
Benj. F. Myers claim	-----	-----	-----	
Placer mining claims in	28	16	10	151.29
Missouri Canyon placer mine	36	16	9	
	30	16	10	19.51
Newark Fluming and Mining Co. placer mine (Greenhorn Creek)	-----	15	9	
		16	10	
Live Oak (½ interest in G. Atkins and J. F. Taylor placer mine)	31	16	10	7.64
Total	-----	-----	-----	663.38

GOLD—LODE MINES.

Ajax Mine. Owner, Miss J. Mitchell, Grass Valley.

Location: Grass Valley District, Secs. 2 and 11, T. 15 N., R. 8 E., 3 miles south of Grass Valley. Elevation 2200′.

This property consists of one patented claim (20 acres) adjoining the Allison Ranch mine on the south. The vein, which strikes north and dips 45° W., has an average width of 1 to 2′ and can be traced for a distance of about 500′ on the surface. It is the southern extension of one of the Allison Ranch veins and occurs in the granodiorite. The shallow drifts and tunnels driven from the banks of Wolf Creek are at present inaccessible.

Alaska Mine. Owner, Alaska Mining Company; M. S. White, Grass Valley.

Location: Grass Valley District, Sec. 1, T. 15 N., R. 8 E., 2½ miles southeast of Grass Valley. Elevation 2500′.

In January, 1916, the mine was being operated by the people who are working the Ben Franklin; J. L. Clayborn, representative. After six months work it was closed again and is idle, September, 1918.

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USGS PP 772

(Ancestral Yuba River)

Gold-Bearing Gravel of the Ancestral Yuba River, Sierra Nevada, California

By WARREN E. YEEND

GEOLOGICAL SURVEY PROFESSIONAL PAPER 772

A restudy of the unmined Tertiary placer deposits in a historic hydraulic mining region

Bureau of Land Management
National Training Center
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Phoenix, AZ 85051

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CONTENTS

ILLUSTRATIONS

TABLES

GOLD-BEARING GRAVEL OF THE ANCESTRAL YUBA RIVER, SIERRA NEVADA, CALIFORNIA

By Warren E. Yeend

ABSTRACT

Enormous quantities of unmined gold-bearing gravel are present on the interstream divides between the Middle Yuba River and the North Fork of the American River. This gravel was deposited by a major Paleocene(?) and Eocene river system (ancestral Yuba River) that flowed into a marine basin to the west from a highland to the east. A mining method using extensive systems of water canals and large iron pipes with huge nozzles, termed hydraulic mining, was used to mine cheaply the gold from this thick gravel from 1855 to 1884. In 1884 a court injunction, obtained by the agricultural interests in the Central Valley of California to prevent the dumping of the mined gravel into the modern rivers, halted hydraulic mining. Owing to this injunction large quantities of gold-bearing gravel remain.

The gravel was restudied with geochemical and geophysical techniques in order to learn more about the occurrence, distribution, characteristics, source, and resource of the placer gold. It is hoped that this new knowledge will encourage the development of economic methods for exploiting the enormous resource believed to remain.

The channel of the ancestral Yuba River and its tributaries cuts into bedrock that is composed of low-grade metamorphic rocks such as Paleozoic phyllite and slate of the Calaveras and Shoo Fly Formations and Mesozoic igneous rocks, primarily granodiorite. Gold-bearing milky-white quartz veins intruding the metamorphic rocks and most prevalent in and adjacent to serpentines served as the primary source for the gold in the gravel.

The gravel, partly covered by the remains of an extensive sheet of volcanic rocks, crops out at a number of places on the broad ridge crests between the major river canyons of the Middle and South Yuba, Bear, and North Fork of the American Rivers.

The gravel is divisible into a lower and an upper unit on the basis of lithology and texture. The lower gravel contains abundant cobble-size clasts, boulders to 10 feet in diameter near the base, many rock types—primarily slate, phyllite, greenstone, granodiorite—and most of the gold. The lower gravel is 70–140 feet thick. It is commonly blue gray and was referred to as the "blue gravel" by miners. The blue-gray color is due to the presence of unoxidized slate and phyllite clasts preserved below the water table.

The upper gravel is distinguished from the lower by an abundance of silt and clay beds, by the presence of clasts predominantly of milky-white quartz and quartzite, and by its low gold content. A leaf flora from clay beds within the upper gravel is correlative with the Chalk Bluffs flora that has been dated as late early Eocene.

Three holes were drilled in the North Columbia diggings in 1968 to obtain samples of an entire section of gravel. The gravel thicknesses ranged from 300 to 455 feet. The gravel in the lower 80–100 feet of all three holes contained most of the gold. Rarely do values in the lower gravel exceed $1.00 per cubic yard ($35.00 per oz); the richest interval was a 2-foot section that contained gold equal to $6.35 per cubic yard. Most of the gravel above the lower rich zone contains gold in amounts generally worth less than $0.02 per cubic yard. Values were at a maximum on bedrock for two holes and at a position 12–15 feet above bedrock for the third hole. Average gold values obtained in the 1968 drilling project were approximately 50 percent of those obtained from drilling by private interests in the same area in 1939. Most of the particles of gold in the lower gravel are 1–2 mm in diameter and 0.1–0.2 mm thick. Gold coarser than 1 mm in diameter was not observed more than 80 feet above bedrock in the drill holes.

Published and unpublished reports containing information on gold values derived from production records and drilling (sampling) in the area give the following averages: Lower gravel, $0.59 per cubic yard, upper gravel, $0.13 per cubic yard.

Four rotary drill holes on San Juan Ridge penetrated 500–650 feet of volcanic breccia above sand, clay, and minor amounts of fine gravel. Gold values from the prevolcanic sedimentary deposits were less than $0.01 per cubic yard. It appears that all four holes were drilled parallel to and along the margin or flood plain of the channel as it passes beneath the volcanic rocks on San Juan Ridge.

It is estimated that within the area studied the exposed parts of the ancestral Yuba River channel contain gold valued at about $188,100,000 distributed in 977,400,000 cubic yards of gravel, the equivalent of an average value of $0.193 per cubic yard. More than 75 percent of this total resource is contained in the vast deposit between Malakoff and Badger Hill Diggings.

On the basis of heavy mineral suites derived from the drill samples obtained in the North Columbia diggings, the gravel can be separated into four zones: (1) a lower zone of ilmenite, zircon, pyrite, amphibole, epidote, and chlorite; (2) a lower middle zone of ilmenite, zircon, pyrite, siderite, amphibole, and epidote; (3) an upper middle zone of ilmenite, zircon, pyrite, and siderite; and (4) an upper zone of ilmenite and zircon. All of these minerals can be derived from local bedrock sources. The zonation reflects differences in degree of physical and chemical weathering on the surrounding slopes. Physical weathering was dominant during the early history of the channel development, and chemical weathering became prominent during the late aggradational stage of the river. Pyrite and siderite are secondary minerals because they were formed after deposition of the gravel. Abundant vegetation trapped

and buried during deposition of the gravel together with the ubiquitous iron-bearing minerals, provided an ideal environment for sulfide development.

The drainage pattern of the ancestral Yuba River and its tributaries can be reconstructed by using existing gravel outcrops, water-current indicators such as imbrication and cross-bedding, and bedrock elevations on the channel floor. Tributaries of the ancestral Yuba River extend beyond the mapped area to the north, east, and south. Original gradients of 17–65 feet per mile are obtained for the ancestral Yuba River drainage upon removing effects of postgravel faulting along the channel and postgravel tilting of the Sierra Nevada. The gradient of the northwest-trending segment from Little York Diggings to North Columbia probably was not affected by tilting, as it would parallel the tilt axis. A gradient of 17 feet per mile seems reasonable for this stretch. From North Columbia to Smartville, the gradient probably was originally 20–25 feet per mile, greater than for the stretch above because the river was flowing perpendicular to the regional slope west of North Columbia. The upstream parts, American Hill to North Columbia and Liberty Hill to Little York Diggings, flowing down a highland front, had gradients of 60–65 feet per mile. The direction of flow from Little York Diggings to North Columbia is interpreted as parallel to the base of this highland.

By comparing known characteristics of the ancestral Yuba River with modern river systems in the Western States, the unknown parameters of drainage basin size and river length can be postulated. Such a study reveals that the ancestral Yuba River may have had a drainage basin as large as 2,000 square miles above the area studied and a length of 150 miles. This would put the headwaters no farther east than western Nevada.

The early Tertiary history of the area was characterized by active river downcutting, high relief, and physical weathering predominating over chemical weathering. Gold was continually being supplied to the rivers but was not transported beyond the drainage basins like the bulk of the rock detritus in the rivers. As the rivers continued to downcut, predominantly during times of flooding and vigorous runoff, the 50- to 100-foot-thick veneer of coarse gravel flooring the river valley would be moved downvalley only to be replaced by new material from upstream. By middle or late Eocene time, the rivers had extended their basins eastward, the steep slopes of the earlier landscape had given way to gentle slopes, and chemical weathering played the prominent role in the breakdown of the rocks. Because of the intensity of the weathering, only the most resistant minerals and rocks survived the journey to the rivers. The valley fill increased in thickness, the rivers continued to aggrade, and extensive flood plains were formed. The climate was probably similar to the present climate on the lower slopes of the Sierra Madre in the State of Vera Cruz, Mexico.

A date of 37.9 ± 1 m.y. was determined by potassium-argon method on biotite from a volcanic tuff lying above the gold-bearing gravel section near North Columbia. A few of the older prevolcanic channels were filled with the volcanic detritus, and a thin veneer of clastic rocks covered parts of the old land surface.

Widespread volcanic mudflows of andesitic composition covered most of the surface during the Miocene and Pliocene. Between successive mudflows, incipient river systems were born, and waterworn boulders and cobbles of andesite were deposited in restricted channels. A rhyolitic biotite-rich tuff dated 8.7 ± 0.5 m.y. is present near the town of Alta.

Probably in late Pliocene time, as volcanic activity subsided, the Sierran block was uplifted and tilted toward the west. During and following the uplift, much of the volcanic cover was eroded and vast areas of the prevolcanic surface exhumed.

Colluvial deposits accumulated near the base of the volcanic cliffs in Pleistocene and recent time because clays in the upper gravel and volcanic section repeatedly failed.

Operators of recent small-scale placer mines along the ancestral Yuba River have been hard pressed to make a profit. Of three mining ventures operating at various times from 1966 to 1970, one remains in operation.

INTRODUCTION

A restudy and evaluation of known gold-producing districts, together with the search for new gold deposits, was initiated in July 1966. The Tertiary gravel in the foothills of the Sierra Nevada in California, from which 14,500,000 ounces of gold was produced, was selected as a promising deposit for restudy. Because of the court injunction of 1884, which virtually put a stop to the hydraulic mining industry, large quantities of gravel remained unmined.

PURPOSE AND OBJECTIVES

A large resource of gold remains in the unexploited alluvial gravels of the ancestral Yuba River owing to an inability in earlier times to extract the gold without degradation of the downstream environment. Assessment of this untapped resource, and, hopefully, the development of acceptable techniques to extract the gold were the main foci of this study. The amount, distribution, and physical character of the gold in these placers had not been fully known, and the bedrock source of it needed additional study. These deposits record an important chapter in the history of the Sierra Nevada region. The deposits constitute an unevaluated potential source of construction material and water resources for nearby growing urban areas. Thus, this study in its broader context provides a geologic framework for the evaluation of the land and mineral resources occurring therein.

METHODS AND CONCLUSIONS

Mapping was done directly on U.S. Forest Service aerial photographs (1:20,000, 1948, 1955, 1966), and the contacts were transferred to topographic quadrangle maps (1:24,000 and 1:62,500). Surface samples were screened and panned in the field, and the panned concentrates transferred to the laboratory for detailed study. In addition to noting textures and lithologies of the gravel where exposed, current indicators (cross-bedding and pebble imbrication) were measured and plotted. Fieldwork was carried on during the fall months of 1966, 1967, and 1969 and during the spring months of 1968 and 1969. In addition to myself, Donald W. Peterson worked on the project from July 1966 to July 1968.

In addition to the conventional geologic field methods of mapping and sampling, many geophysical techniques

were employed in an effort to learn more about the subsurface characteristics of the gravel. The techniques used include seismic refraction, ground and airborn magnetic and electromagnetic (EM) surveys, gravity and resistivity surveys, and induced polarization. The geophysical field studies were conducted during the summers of 1967 and 1968 under the supervision of H. W. Oliver.

During the fall and winter of 1968 and January 1969, rotary and churn drilling was done to check the validity of the geophysical interpretations and to obtain samples from the gravel for determination of gold values and for heavy-mineral study. The results of that drilling are presented here.

In 1968 the U.S. Bureau of Mines, under the guidance of the U.S. Geological Survey, selected the Badger Hill hydraulic pit as a focus area. A rather thorough study of the old hydraulic diggings ensued that involved many of the branches of the Bureau of Mines, including the mining research group from Denver, Colo. Extensive geophysical surveys were made, numerous holes were drilled with both rotary and large bucket drills, and large samples were taken for the purpose of concentrating gold and other heavy minerals. It was originally hoped that efficient mining techniques might be developed to extract gold and other metals from the gravel at this locality. The "mining research" phase of the project, however, did not progress beyond the sampling program.

The study has revealed that the early work of Lindgren (1911) was quite complete and thorough in locating and describing the gold-bearing gravel.

The major remaining placer gold resource appears to be on San Juan Ridge between Badger Hill and Malakoff State Park. A liberal estimate indicates that 800,000,000 cubic yards of gravel at this locality could contain recoverable gold worth $140 million ($35.00 per oz). Drilling in the North Columbia diggings in 1968 produced samples that rarely contain gold value in excess of $1.00 per cubic yard; this value is considerably less than obtained from earlier drilling in 1939 by private interests. Estimates of total volume and values of unmined gravel in the entire area studied indicate that gold worth $188 million ($35.00 per oz) could be present. This estimate does not include the gold in gravel beneath the thick (500–700 ft) volcanic breccia on San Juan, Washington, and Harmony Ridges.

Vertical distribution of heavy minerals within the gravel implies an early history of physical breakdown of the source rocks with limited chemical weathering. Most of the gold, in highest concentration in the lower 80 feet of immature gravel, was supplied to the river during its early downcutting stage. The river in later Eocene time was characterized by a low gradient, a wide flood plain, large meanders, and a sediment load indicative of intensive chemical weathering. During this time, gold either was not carried into the rivers or was flushed into the marine environment to the west.

MINING HISTORY

Placer gold first discovered and mined along the deep valleys of the youthful streams and rivers in the Sierran foothills was, in time, traced to sources high above the streams on the drainage divides. What we now call the "gold-bearing Tertiary gravel" was found to contain large amounts of gold, but in relatively low concentrations. The simple small-scale mining techniques employing a pan, a sluice box, or a rocker, used by the lone miner along the present streams, gave way to the land-destroying methods of hydraulic mining. First used in 1852 in Yankee Jims in Placer County and near Nevada City (Clark, 1965), it was during the 29 years from 1855 to 1884 that the great bulk of gravel was washed by this simple but devastating method. Nozzles with openings as large as 9 inches in diameter directed water with enormous force at the gravel beds, breaking apart the generally poorly cemented conglomerate. Large quantities of gravel could be washed with little effort by playing the jet of water back and forth across the gravel exposures. The manmade rivers carried the great bulk of the sand and gravel into the natural drainages, while the gold was trapped in the riffles and sluices, which were often heavily charged with mercury. The natural drainages carried most of this displaced gravel to the edge of the Central Valley, where, because gradients in river channels were lower, it was deposited, filling and choking the channels. Flooding and silting repeatedly destroyed vast amounts of farmland and frequently threatened to destroy property in the large cities of Marysville and Sacramento. Litigation between the miners and farmers continued more than 10 years, culminating in the famous court case Woodruff vs. North Bloomfield Gravel Mining Co. (Kelley, 1959, p. 237–240). Judge Lorenzo Sawyer issued an injunction in 1884 against the company, now often referred to as the "Sawyer decision," that prohibited the dumping of debris into the Sacramento and San Joaquin Rivers and their tributaries. A precedent was set, and hydraulic mining nearly ceased. Some companies constructed debris dams and made a pretext of storing the washed gravel, but the big hydraulic mining effort was finished.

Drift mining along the gravel-bedrock contact continued after the shutdown of the hydraulic mines, but only the richest gravel was mined. Gilbert (1917) estimated that 30 million cubic yards of gravel was mined by drifting.

In an attempt to revive the hydraulic mining industry, the California Debris Commission was created in

1893 to aid and oversee the mining. This effort proved unsuccessful. and with the inflation following World War I. gold production dropped further. In 1930 public funds were allotted for the construction of large debris dams on the major rivers draining the foothills. Englebright dam on the Yuba River and North Fork dam on the North Fork of the American River were built for storing hydraulic tailings. The California Debris Commission obtained assurances from the mining people that the dams would be used for these purposes and that the costs of construction would be repaid by the mining industry. However. hydraulic mining never again flourished, and less than 3 percent of the storage capacity of the dams was utilized. What little "gold fever" remained in the hydraulic mining industry was discouraged by the high costs of water and equipment, and by increasing governmental regulation. Even the increase in gold price from $20.67 to $35.00 per fine ounce in 1934 did not provide the economic incentive to revive hydraulic mining.

Following the shutdown of the hydraulic mines in 1884. most of the California gold was produced by floating bucket dredges. which operated in the vast gravel deposits along the major drainageways at the junction of the mountain front and the Central Valley.

Pine trees and manzanita bushes have since sprung up in the hydraulic pits. Most of the old access roads are overgrown. except where frequented by the bottle collector. hunter. motorcyclist. or trash disposer. Little remains in the way of mining artifacts. Even the old rusty hydraulic pipes are gone; they were carted off for road culverts.

AREA OF STUDY

The area of study is roughly that part of the Sierran foothills between the North Fork of the American River and Middle Yuba River (fig. 1). The mapped area comprises parts of the Emigrant Gap. Alleghany. Colfax. and Nevada City quadrangles (1:62,500 scale). It contains a part. but not all. of the ancestral Yuba River drainage basin. This particular area was selected for study because: it contains the largest remaining deposits of Tertiary gold-bearing gravel in the Sierra Nevada: a significant portion of the total Tertiary placer gold produced from California has been derived here: a fairly large gold resource was believed to still exist: and exposures and access are relatively good.

PREVIOUS WORK

The comprehensive report by Lindgren (1911) stands as a classic in its complete and thorough coverage of the Tertiary gravels of the Sierra Nevada. I have drawn extensively from this work because much of the information that was available in Lindgren's day. particularly concerning gold values. has vanished. Whitney's discussion of the gravel (1880) is of interest because it was written during the height of the mining activity. A rather exhaustive quantitative study of the hydraulic mining debris that was made by Gilbert (1917) is useful when figuring average gold values. Early folio maps by Lindgren (1900) and Lindgren and Turner (1895) show the distribution of Tertiary gravel as well as bedrock (scale 1:125,000). The study area is covered by the more recent map compilation of Burnett and Jennings (1962). Good discussions of the geology of the gravel and information on potential gold resources are in Jenkins (1946), Averill (1946), Haley (1923), Hammond (1890), and Turner (1891). The Jarman (1927) report on the feasibility of resuming hydraulic mining contains a rather thorough summary of unmined gravel quantities. A report on the fossil flora contained in the gravel (MacGinitie, 1941) is still our best evidence for establishing age and climatic conditions of deposition of at least the upper part of the gravel. Clark's reports (1965. 1970) are handy useful references on the gravel and its occurrence throughout the Sierra Nevada. An informative up-to-date summary of the regional geology of the Sierra Nevada, including a discussion of both the bedrock and overlying rock units, is given by Bateman and Wahrhaftig (1966). An early publication derived from the studies by the Geological Survey concerns the area on San Juan Ridge. particularly the gold resource and geophysical investigation of the gravel (Peterson and others. 1968).

ACKNOWLEDGMENTS

I am grateful to local property owners who permitted access to their land and who cooperated in various phases of the work done on their private property: Herbert Jeffries. William Coughlan. Harold Helland. D. R. Schiffner. M. J. Meredith. and San Juan Gold Co. San Juan Gold Co. was most helpful in providing access to unpublished maps and reports. John H. Wells, Bureau of Land Management. provided practical instruction in gold recovery and concentration. Mr. and Mrs. August Ebbert. local residents, provided much needed aid. assistance. and logistic support to everyone associated with the project: their generosity knew no bounds. The California Division of Beaches and Parks. the State Division of Forestry. and the U.S. Forest Service were cooperative whenever called upon.

GENERAL GEOLOGY

Unmined gold-bearing gravels are mapped on plate 1. which also shows the distribution of the overlying volcanic rocks and adjacent underlying metamorphic and igneous rocks. Only those areas known or believed to contain gold-bearing gravel were mapped for this study.

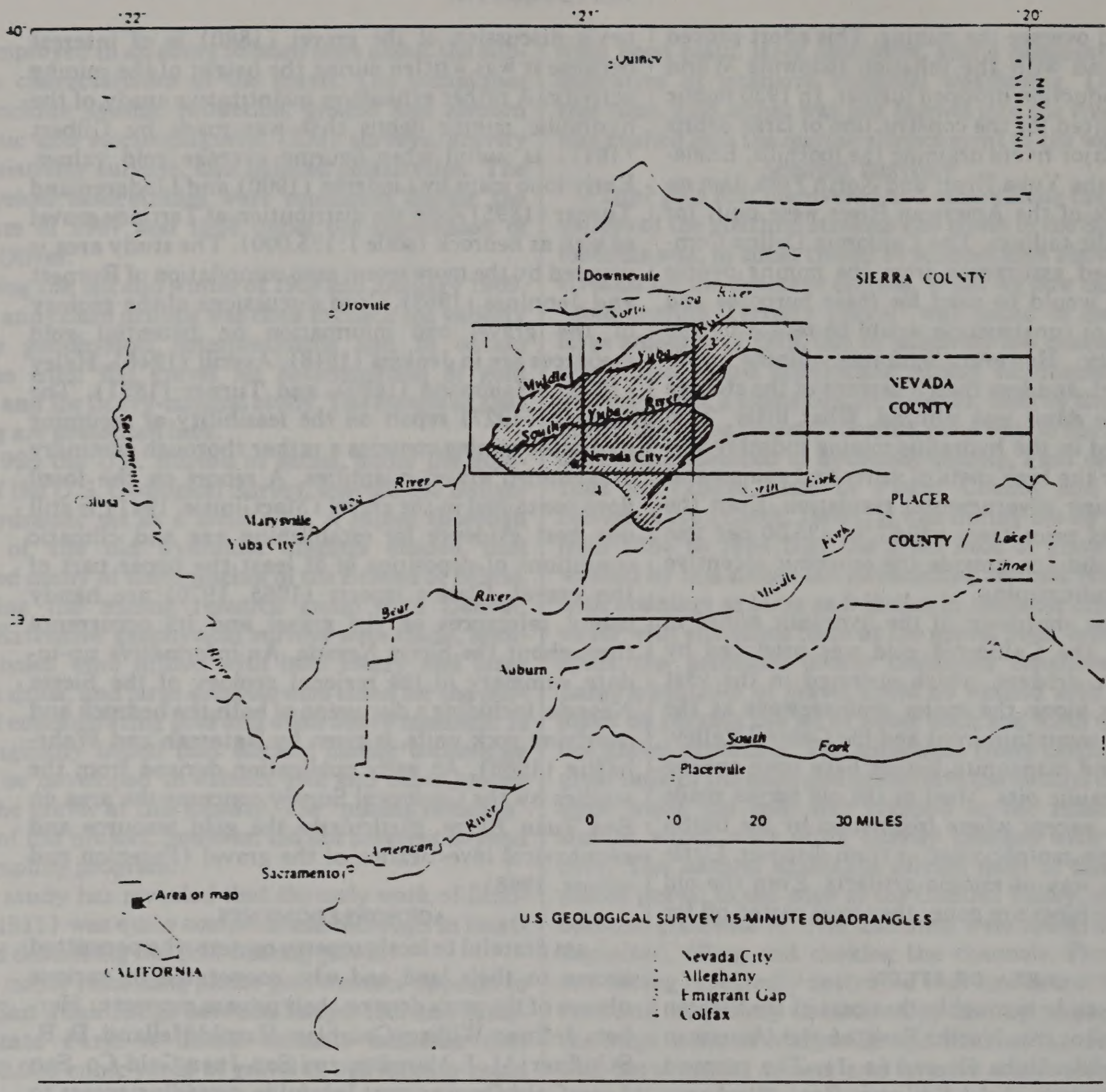


FIGURE 1.—Index map showing location of study area (patterned).

BEDROCK

Two types of bedrock are shown on the geologic map: igneous and metamorphic. The distribution of serpentines and of some quartz veins is shown in figure 2.

The metamorphic rocks are all low grade. They occur in a wide north-west-trending belt across the central part of the mapped area. Commonly referred to as the Calaveras Formation (Lindgren, 1900; Ferguson and Gannett, 1932) of late Paleozoic age, they consist mostly of fissile slate, siliceous phyllite, hornfels, quartzite, and greenstone. In the eastern part of the area are dark-gray slates of the Silurian(?) Shoo Fly Formation (Clark and others, 1962, p. B16, B17). Metamorphism has been so slight that many of the metasedimentary rocks still show well-preserved sedimentary textures and structures such as crossbedding, graded bedding, lenticular and wavy bedding, and alternating silt and clay beds. The slates and phyllites are locally rich in pyrite. A typical phyllite contains detrital quartz and feldspar in a groundmass of quartz, sericite, and chlorite. Mesozoic metavolcanic rocks typical of the western Sierra metamorphic belt (Bateman and Wahrhaftig, 1966, p. 113) occur in the extreme western part of the area.

Mesozoic igneous rocks intrude the metamorphic rocks throughout the area. Biotite-hornblende granodiorite constitutes the two largest intrusive bodies in the eastern and western parts of the area. Smaller intrusive bodies consist of diorite, serpentine, gabbro, and diabase. A saussuritic granitic rock rich in epidote

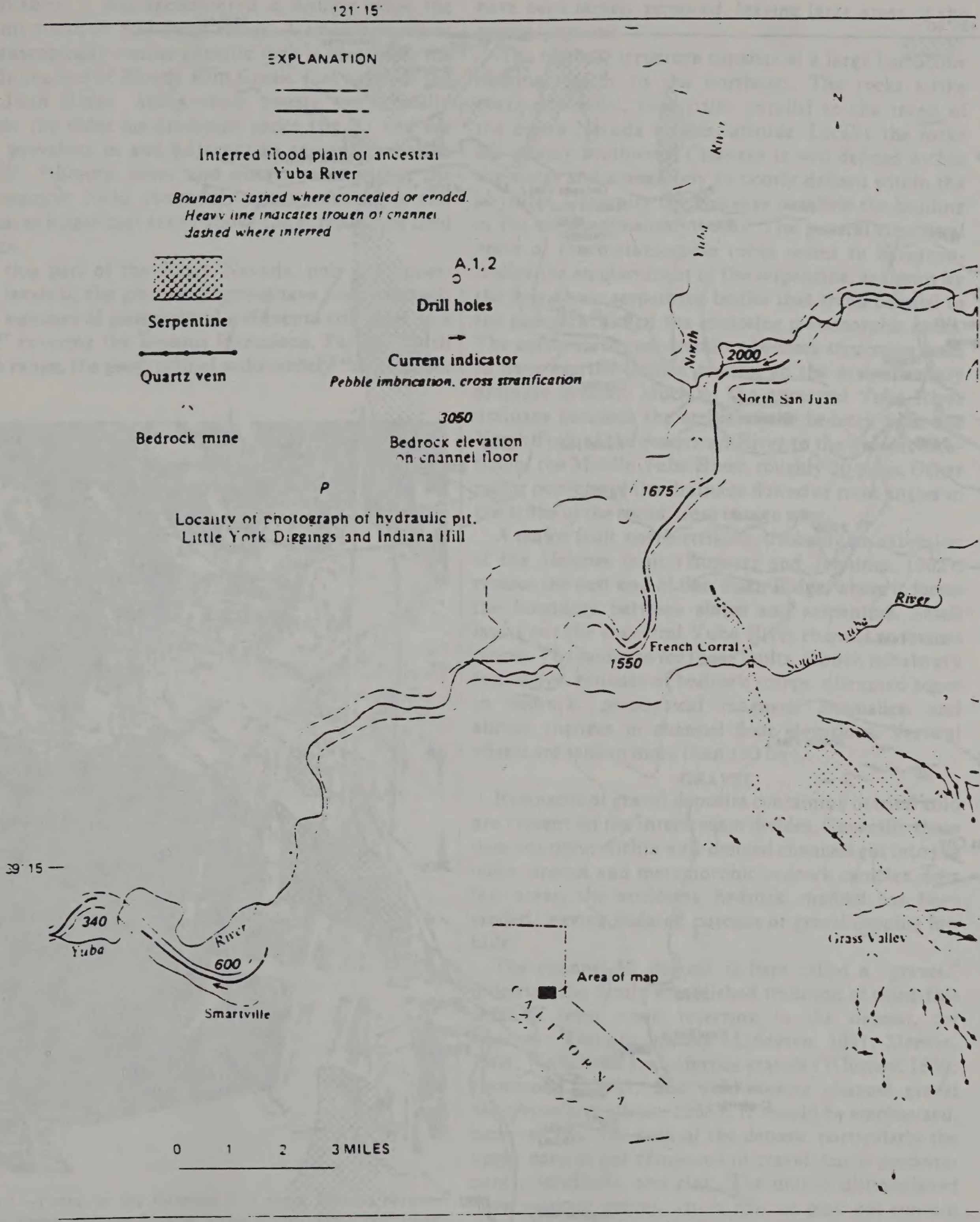
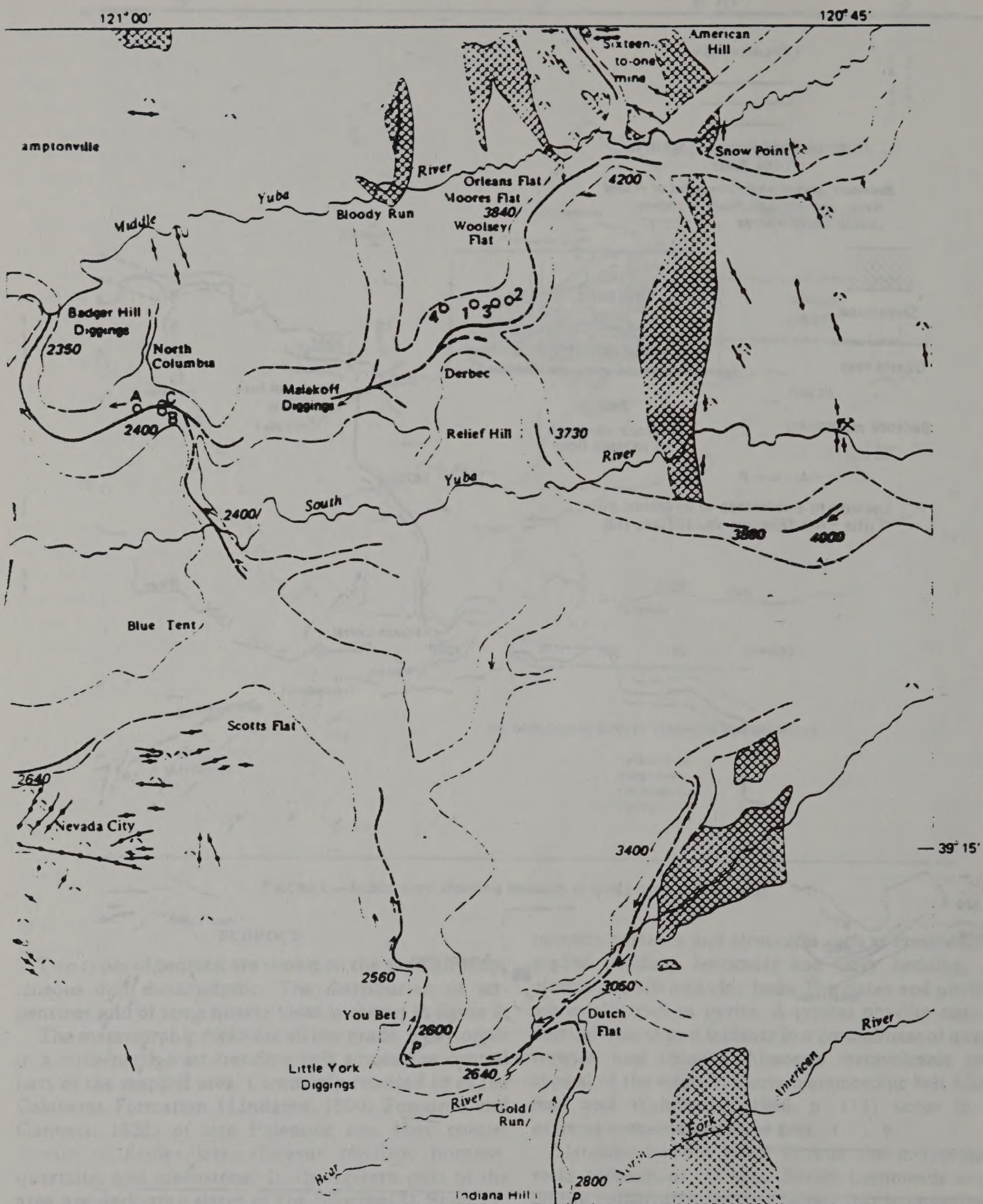


FIGURE 2.—Trace of part of ancestral Yuba River



drainage. Sierra. Nevada. and Placer Counties. Calif.

and allanite(?) was encountered in drilling below the volcanic rocks on San Juan Ridge. A small outcrop of a megascopically similar granitic rock is present in the middle reaches of Bloody Run Creek, just north of the San Juan Ridge. Milky-white quartz veins locally intrude the older metamorphic rocks (fig. 3) and are most prevalent in and adjacent to serpentine bodies (fig. 2). Siliceous zones and quartz veins within the metamorphic rocks commonly resist weathering and remain as ridges that stand above the surrounding land surface.

In this part of the Sierra Nevada, only the uppermost levels of the granitic plutons have been exposed: large volumes of geosynclinal sediments still exist as a "roof" covering the igneous intrusions. Farther south in the range, the geosynclinal sedimentary "roof" rocks have been largely removed, leaving large areas of the plutons exposed.

FIGURE 3.—Portal of the German Bar mine, located between Snow Point and the Middle Yuba River. Slate and greenstone of the Calaveras Formation are intruded by gold-bearing(?) quartz, a rock type common in prevolcanic gravel, and the source for much of the gold contained in the gravel.

The regional structure consists of a large homocline dipping steeply to the northeast. The rocks strike north-northwest, essentially parallel to the trend of the Sierra Nevada at this latitude. Locally the rocks dip steeply southwest. Cleavage is well defined within the slates and moderately to poorly defined within the phyllites. Generally the cleavage parallels the bedding in the metasedimentary rocks. The general structural grain of the metamorphic rocks seems to have controlled the emplacement of the serpentine, as shown by the long linear serpentine bodies that trend parallel to the general strike of the enclosing metamorphic rocks. The north-northwest-trending bedrock structures seem to have exerted limited control on the early Tertiary drainage system. Much of the ancestral Yuba River drainage parallels the strike of the bedrock from the North Fork of the American River to the present location of the Middle Yuba River, roughly 20 miles. Other major portions of the drainage flowed at right angles to the strike of the rocks, from east to west.

A major fault north-striking, probably an extension of the Melones fault (Burnett and Jennings, 1962), crosses the east end of San Juan Ridge, where it forms the boundary between slates and serpentine. Small faults cut the ancestral Yuba River channel at several places. The evidence for these faults, though not always conclusive, consists of bedrock scarps, disrupted zones in bedrock, geophysical magnetic anomalies, and abrupt changes in channel floor elevations. Vertical offsets are seldom more than 100 feet.

GRAVEL

Remnants of gravel deposits containing detrital gold are present on the interstream divides. Generally these deposits occur within well-defined channels cut into the older igneous and metamorphic bedrock complex. In a few areas, the enclosing bedrock channel has been eroded, leaving isolated patches of gravel capping low hills.

The channel-fill deposit is here called a "gravel," following the firmly established tradition of using this textural term when referring to the deposit, for example: Tertiary gravels (Lindgren, 1911; Merwin, 1968; Wells, 1969), auriferous gravels (Whitney, 1880; Hammond, 1890), and gold-bearing channel gravel (Peterson and others, 1968). It should be emphasized, however, that the bulk of the deposit, particularly the upper part, is not composed of gravel, but is predominantly sand, silt, and clay. The unit is distinguished from younger gravel, which may or may not contain detrital gold, by the absence of unmetamorphosed volcanic clasts. Younger gravel contains abundant clasts of rhyolite, andesite, and their weathering products.

Such vast amounts of gravel were removed by hydraulic mining that at some places bedrock is now exposed where thick gravel deposits formerly lay. In other localities, for example, North Columbia and the southern part of the Badger Hill pit, large amounts of gravel remain. The hydraulic pits are outlined on the geologic map (pl. 1), and the areas where gravel was stripped all the way to bedrock are shown.

Although many of the gravel exposures have been isolated by erosion, the major drainage system can be reconstructed with reasonable certainty. (See section entitled "Trace of Restored Channel.")

LOWER AND UPPER CONTACTS

The basal gravel rests with pronounced angular unconformity on the steeply dipping slate and phyllite. Where the lower contact is well exposed, as on the floor of hydraulic pits, the bedrock surface is smoothed, polished, grooved, and fluted (fig. 4). Both igneous and metamorphic bedrock show the typical erosional features associated with active river scour. The old erosion surface is generally intact on the granitic bedrock. In metamorphic bedrock, the surface was commonly broken up and destroyed during mining. Rock cleavage, joints, and associated fractures that are more common within the slates and phyllites acted as natural traps for the lodgment of detrital gold particles. This fact was clearly evident to the early miners, who typically would tear up 1–5 feet of the old channel floor in their search for the entrapped gold. The granitic surface, generally smooth, presented a poor environment for the lodgment of gold particles.

Outside the pits, the precise position of the lower contact at the margin or lateral boundary of the channel is generally obscured by thick soil, and subtle evidence had to be used to locate the contact. In some places a topographic break indicates the contact position. Elsewhere, resistant rounded alluvial quartz pebbles indicate underlying gravel, and angular siliceous rock chips in the soil signify underlying bedrock (figs. 5, 6). The channel rim or edge, as evidenced by the contact of gravel and bedrock, could generally be correctly mapped to within 50 feet. Cuts showing weathered bedrock below the gravel were observed only near the margins of the old river channels. At these localities the weathered bedrock was preserved by the overlying gravels. It seems doubtful that weathered bedrock could survive active fluvial scour, as would have been common near the center of the river channel.

The rocks overlying the gold-bearing gravel have only one unifying characteristic: they are rich in fresh

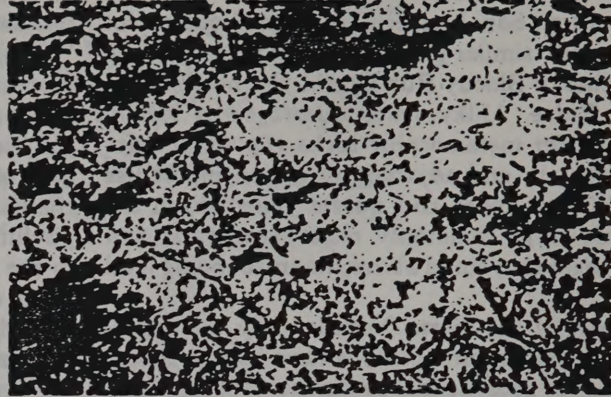


FIGURE 5.—Surface of soil developed on phyllitic bedrock. Note angular bedrock chips remaining as a lag on soil surface.

FIGURE 4.—Fluted and scoured granitic bedrock floor of the once gravel-filled channel exposed in a hydraulic pit near Peterson's Corner 2 miles southwest of North San Juan. The overlying gravel was removed by hydraulic mining in the late 1800's. The river flowed toward the observer.

FIGURE 6.—Surface of soil developed on gold-bearing gravel. Note subrounded pebbles of quartz and siliceous phyllite weathering out of the alluvial gravel. Such subtle evidence as differences in degree of roundness or angularity (compare fig. 5) is used in locating gravel-bedrock contacts. Eyeglass case indicates scale.

volcanic material. These younger rocks include: bentonite and tuff beds at North Columbia; volcanic cobble gravel at Scotts Flat, Chalk Bluffs, and Malakoff Diggings; andesite breccia at Woolsey Flat; and colluvium at numerous localities.

In contrast to the overlying rocks, the upper part of the auriferous gravel lacks volcanic detritus and is everywhere characterized by fine-grained alluvial material such as sand, silt, and clay. In places where tuff and bentonite overlie the gravel, the contact seems gradational, and it does not look like erosion took place between the deposition of the two units. Where the gold-bearing gravel is overlain by material such as volcanic cobble beds, andesite breccia, or colluvium, the contact is quite clearly an erosional unconformity.

LITHOLOGY AND TEXTURES

Close examination suggests that the gold-bearing gravel can be divided into an upper and a lower unit. Although the units are lithologically and texturally distinct, no distinct contact can be selected. Consequently, the gravel was mapped as a single unit. For text description, however, the gravel is divided into two units.

Pebble counts were made at selected localities along the channel in both the upper and lower gravel units. At similar sampling points, percentages of the different size clasts were measured by screening and weighing the various size fractions. Field descriptions of measured gravel sections at Woolsey Flat, Malakoff Diggings, North Columbia, Badger Hill, and Chalk Bluffs are given with sections on plate 2 and figures 7 and 8. Descriptive gravel sections of the drill holes on San Juan Ridge and in the North Columbia hydraulic pit are shown on plate 2.

Results of the studies of the heavy minerals and detrital gold obtained from the gravels are discussed under the sections "Placer Gold" and "Other Heavy Minerals."

LOWER GRAVEL

The lower gravel contains most of the gold and consequently was the unit most sought by the early miners. Relatively few exposures of the lower gravel can be found, as it was either completely mined out or, because of the thick cover of overlying gold-poor upper gravel, was never exposed. Localities where the lower gravel could be seen in 1968 include the hydraulic pits at French Corral, Birchville, Badger Hill, Moore's Flat, Sailor Flat, Little York, Dutch Flat, and the unnamed hydraulic pit 1 mile northeast of Edwards Crossing near Spring Creek. Even fewer exposures can be found of undisturbed lower gravel resting on bedrock. One such exposure is near the northwest edge of the hydraulic pit at Moore's Flat (fig. 9). Granitic boulders more than 6 feet in diameter are not uncommon at this locality, and some are as much as 10 feet in diameter. These boulders must have been transported at least 8 miles, the distance to the nearest upstream bedrock exposure of similar porphyritic granite. Large boulders 6–10 feet in diameter lie along the south side of the Malakoff pit (fig. 10). The boulders probably have been disturbed by mining activities and are not in place; but as they probably were not moved very far, they do give some idea of the maximum boulder size in the lower gravel. These rocks are siliceous phyllite similar to the bedrock in the Malakoff area. At Badger Hill and French Corral, the lowest gravels are not exposed, and the clasts are smaller than those at Malakoff and Moore's Flat. Most of the lower gravel unit is exposed at these localities and is predominantly of cobble size. At Badger Hill, good exposures of the lower gravel unit are seen in the walls of the lower hydraulic pit (fig. 11). Imbrication within the cobble gravel is moderately well developed and indicates a south-to-north current flow.

Size analyses of 26 samples of lower gravel collected throughout the area give the following averages (using Wentworth's size classification): cobble and boulder (>64 mm), 13 percent; pebble (64 to 4 mm), 56 percent; granule and sand (4 to $\frac{1}{16}$ mm), 28 percent; and silt and clay (<$\frac{1}{16}$ mm), 3 percent. Because of the great variability in texture of the lower gravel both vertically and laterally along the channel, these figures may be meaningless when applied to any one area. However, they can be useful in a general comparison of texture of the total unit with the textures of the upper gravel. Sorting in this lower unit, as would be expected in a coarse gravel, is poor.

Pebble counts of the 16- to 32-mm size fraction give the following averages from 26 samples: bluish-black siliceous slate, 52 percent; weathered igneous rocks, 31 percent; milky quartz, 12 percent; other, 5 percent. The compositionally immature nature of the lower gravel indicated by the pebble count is substantiated by the heavy minerals present. Chlorite, amphibole, and epidote are common within the lower gravels, whereas they are conspicuously absent from the upper gravel.

The lower gravel varies in thickness from 70 to 140 feet, depending on where one places the upper boundary. Generally the boundary is placed where pebble and cobble gravel is replaced upward by beds that are predominantly pebble and sand. The lower 130 feet of measured section at Malakoff Diggings probably corresponds to the lower gravel (pl. 2). Most of this part of the section consists of cobbles and pebbles; boulders

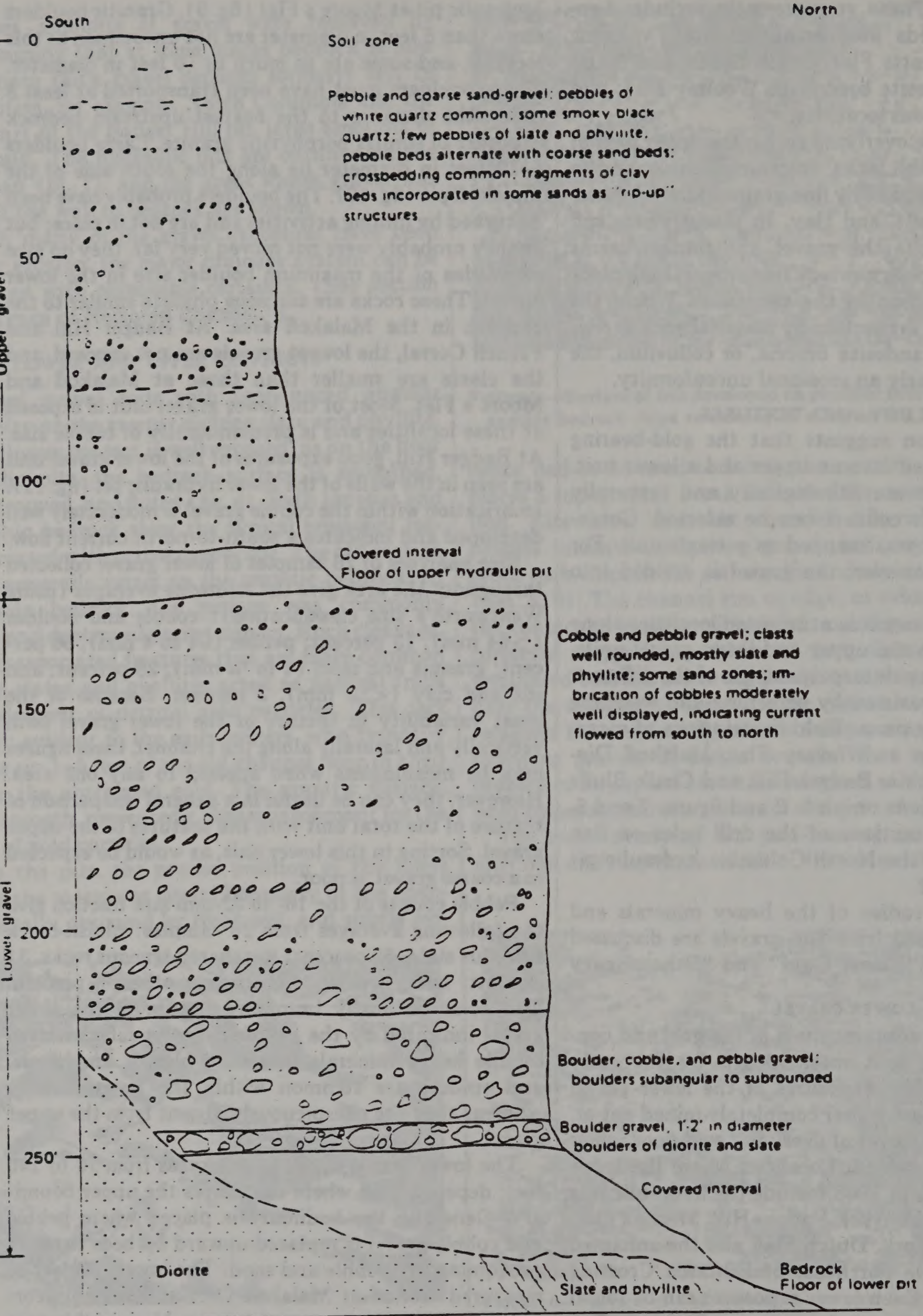


FIGURE 7.—Composite section of gold-bearing gravel exposed in the Badger Hill diggings.

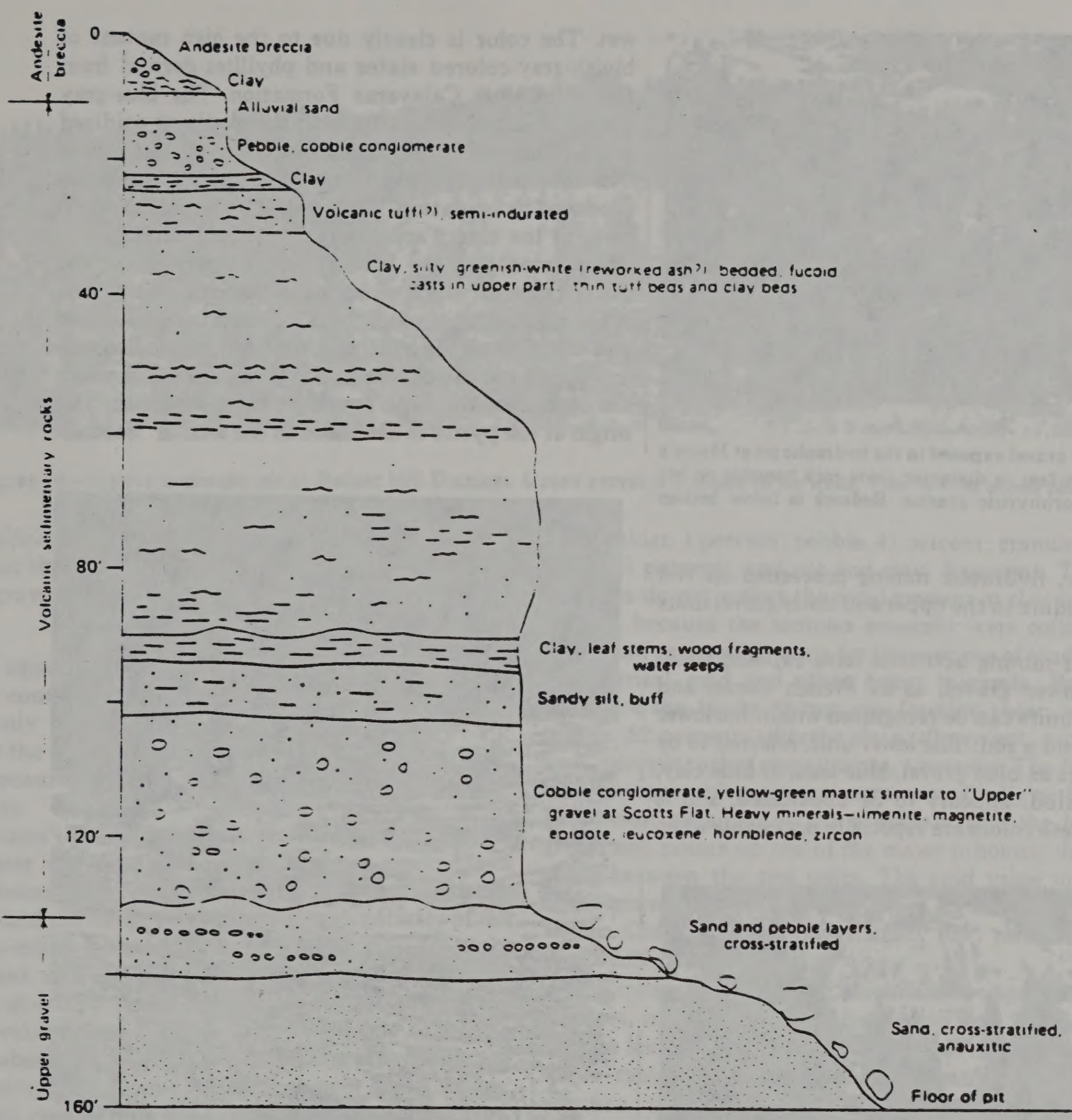


FIGURE 8.—Section measured at Chalk Bluffs showing the change from the upper gold-bearing gravel to volcanic sedimentary rocks and andesite breccia. The section is located on the east side of the large hydraulic pit near You Bet.

appear in only the lowest 30 feet. All exposed gravel measured at Woolsey Flat is in the upper gravel. The lower 75 feet of section is covered but presumably would contain the lower gravel unit (pl. 2). The gravel section measured at North Columbia is part of a composite section shown on plate 2 as hole A. The description of the upper 50 feet and lower 260 feet of this section was obtained from drill-hole data. The description of the middle part of the section (260 feet) was made from outcrops within the hydraulic pit. A great deal more information could be obtained about textures from the actual exposures than from the drill data. The churn drill shattered all the coarse constituents of the gravel to sand size and smaller and made reconstruction of original textures and lithologies exceedingly difficult. On the basis of such drilling characteristics which can be useful indicators of boulders and cobbles, it would appear that the lower 90–100 feet of gravel drilled is of the lower gravel unit, roughly corresponding to the heavy mineral zones III and IV as discussed in the section "Other Heavy Minerals." The lower gravel at Badger Hill is approximately 140 feet thick.

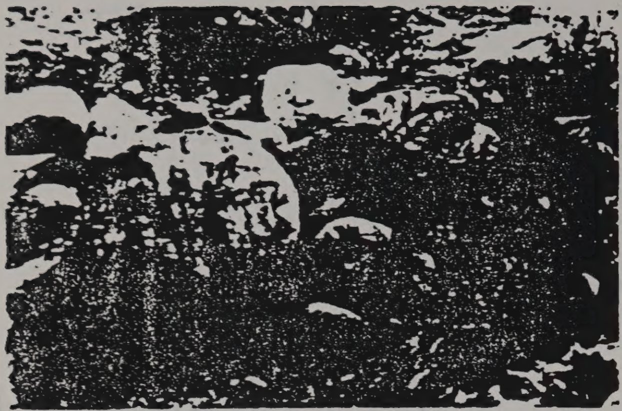


FIGURE 9.—Lower gravel exposed in the hydraulic pit at Moore's Flat. Boulders 6 feet in diameter (note rock hammer on left boulder) are porphyritic granite. Bedrock is below bottom of picture.

At this locality, hydraulic mining proceeded on two levels corresponding to the upper and lower gravel units (fig. 7).

Where recent mining activities have exposed fresh, unweathered lower gravel, as at French Corral and Birchville, two units can be recognized within the lower gravel, a blue and a red. The lower unit, referred to by the older miners as blue gravel, blue lead, or blue clay, is water saturated, appears to be unoxidized, and is bluish gray. These colors are especially noticeable when wet. The color is clearly due to the high content of bluish-gray-colored slates and phyllites derived from the ubiquitous Calaveras Formation. The blue-gray gravel contrasts markedly with the overlying oxidized gravel, the upper unit, sometimes called the red gravel. The differences between the two appear to be those produced by oxidation. The general textures and lithologies of the clasts are similar. As Lindgren (1911, p. 76) pointed out, the blue gravel is a zone below the water table in which reducing conditions prevail and is not a distinct lithologic unit different in composition and origin from the overlying oxidized gravel. Secondary pyrite coats pebbles and is disseminated through the matrix of the blue gravel at many localities. The origin of the pyrite is discussed in the section "Second-

FIGURE 10.—Boulders are as much as 10 feet in diameter near the floor of the Malakoff hydraulic pit. View is north across lake in west end of pit. Boulders are siliceous phyllite similar to bedrock in area.

FIGURE 11.—Lower gravel in the Badger Hill hydraulic pit. View is west near northernmost part of pit. Gravel is exposed on side of pit, and bedrock slopes up steeply beneath gravel. Gravel-bedrock contact is not exposed but probably is 20–30 feet below lowest exposure of coarse gravel. Largest boulders exposed are 3 feet in diameter. Note aluminum notebook for scale.

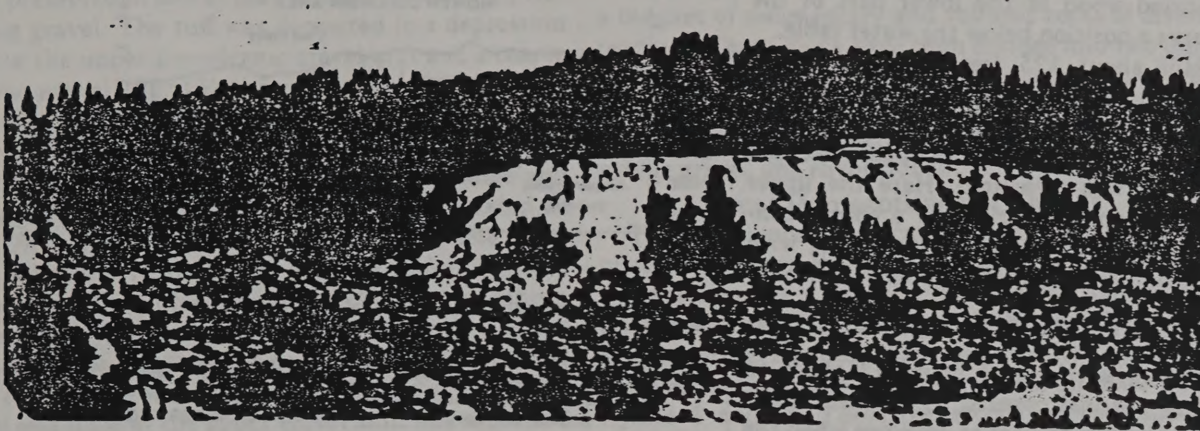


FIGURE 12.—Upper hydraulic pit at Badger Hill Diggings. Upper gravel is exposed in pit walls. View is west across pit.

ary Sulfides." At those places where the water table is within the upper gravel as at North Columbia, all the lower gravel is blue (zones III and IV, pl. 2).

UPPER GRAVEL

The upper gravel, unlike the lower, is exposed more or less continuously along the old river channel. It is commonly exposed in cliff faces and constitutes the bulk of the deposit in most areas. Figure 12 shows typical exposures in the hydraulic pit at Badger Hill Diggings.

Features that help distinguish the upper gravel from the lower are: (1) the generally finer grain, clasts rarely being larger than pebble size; (2) the abundance of clay and silt beds; (3) compositional maturity of the clasts—milky-white quartz and quartzite are very abundant and sand and silt-size grains are predominantly quartz; (4) the heavy-mineral content, almost exclusively zircon, ilmenite, and magnetite.

Crossbedding and cut-and-fill features are common sedimentary structures within the upper gravel. Large, sweeping crossbeds 8 feet across are observed in places. Associated with the finer grained character of this gravel is pronounced bedding and fair to good sorting. Pebble beds are most prevalent in the lower part of the upper gravel, and clay beds are common in the upper part of the section. Commonly the clay is greenish gray and weathers to brown. Thin sand and silt zones, generally less than a foot thick and heavily impregnated with iron oxide, occur sporadically throughout the upper gravels. These zones are possible evidence of old water tables within the gravel. Five such zones were noted in the upper gravel at Malakoff in 325 feet of section (pl. 2).

Size analyses of the upper gravel determined from 35 different localities give the following averages: cobble and boulder, 1 percent; pebble, 41 percent; granule and sand, 53 percent; and silt and clay, 5 percent. These averages do not reflect the total amount of clay in the section, because the samples generally were collected from sand and pebble units for the purpose of studying the detrital gold and other heavy minerals. Pebble counts of the 16- to 32-mm size fraction show: white quartz, 52 percent; siliceous slate, 40 percent; quartzite, 7 percent; other constituents, 1 percent. The figure of 52 percent white quartz contrasts markedly with the 12 percent obtained as the average within the lower gravel and points up one of the major lithologic differences between the two units. The gold value of the upper gravel, as discussed in detail in section "Gold Values Old and New," is generally low, rarely more than $0.02 per cubic yard.

The maximum measured thickness of upper gravel, 400 feet, is in the North Columbia pit (pl. 2). In this pit, the lower 160 feet of the upper gravel is below the water table, and sulfides and carbonized wood are common within it. As the upper gravel at North Columbia grades upward into bentonitic clay and tuff of volcanic origin with no apparent depositional break, the section here probably is more or less complete. The overlying volcanic rocks at North Columbia are thin and of spotty occurrence and could not be mapped at the scales used. For mapping purposes, they have been included in the prevolcanic gravel unit. At Malakoff Diggings (pl. 2) there is at least 325 feet of the upper gravel unit. Many of the clay beds exposed in the hydraulic cliffs have failed and mud and earth flows are common in the pit. Here also some of the clays in the upper part of the section are probably of volcanic origin. At Woolsey Flat (pl. 2) there is approximately 200 feet of upper gravel. The lower boundary is not

exposed. Carbonized wood in the lower part of the upper gravel marks a position below the water table.

At Badger Hill about 125 feet of upper gravel remains (fig. 7). Silicified wood, distributed sporadically throughout the upper gravel, is particularly abundant. At Chalk Bluffs (fig. 8), only the uppermost part of the upper gravel was measured. Here the upper boundary is distinct and is marked by a cobble conglomerate with abundant volcanic clasts resting with erosional unconformity on sand of the upper gravel unit.

CHANNEL CONFIGURATION

The diagrammatic sections of figure 13 show an idealized Tertiary channel covered by volcanic rocks and an example of land surface that develops when the volcanic cover is eroded away. The porous and permeable gravel, once exposed, is more resistant to erosion than the bordering bedrock, be it granitic or slate. At the few localities where the gravel has not been mined, the channel is expressed as a low ridge, a topographic reversal of what originally was a linear depression. Rainwater tends to percolate into the gravel instead of running off and thus causes little physical erosion. The quartz and quartzite rock types common in the upper gravel further resist chemical breakdown. A short stretch of the channel between French Corral and Birchville is a characteristic gravel ridge marking the channel location (fig. 14).

In the North Columbia area, the channel is very broad and the gravel thick; 15 miles downstream, the gravel is much thinner and occupies a narrower channel (fig. 14). The channel configuration at North Columbia is determined from five holes drilled to bedrock across the channel in the early 1900's by private interests. Exposures of bedrock throughout much of the area from Birchville to French Corral allow an accurate depiction of the configuration of the channel floor. An adequate explanation for the marked difference in channel size between the two areas is not apparent. The

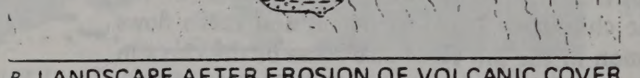


FIGURE 13.—Diagrammatic cross section of an idealized gravel-filled Tertiary channel showing rock relations before, (A), and after (B), erosion of covering volcanic rocks.

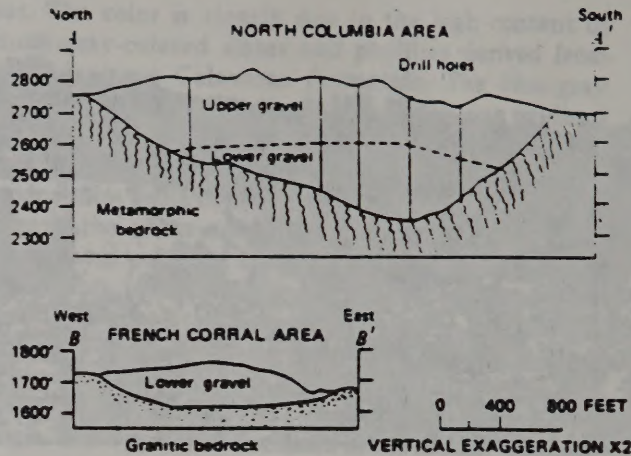


FIGURE 14.—Cross sections of gravel-filled channels near North Columbia and French Corral. French Corral is approximately 15 miles downstream from North Columbia along the ancestral Yuba River. Lines of sections are shown on plate 1.

gravel in the French Corral area belongs to the lower unit; one explanation for the absence of the upper unit is that it has been eroded away. The French Corral area lies farther from the existing volcanic covering rocks than the North Columbia area; most likely the gravel at French Corral has been exposed to erosion for a longer period of time.

AGE

Fossil leaves were found at only one locality within the study area, within clay beds in the lower pit at North Columbia (pl. 2, index map). The composite stratigraphic section measured in the North Columbia area (pl. 2, hole A) shows that the leaf horizon is near the middle of the upper gravel. The flora in this horizon is similar to that described and dated by MacGinitie (1941) as late early Eocene (Bateman and Wahrhaftig, 1966, p. 135). The stratigraphic horizon at Chalk Bluffs, for which MacGinitie named the Chalk Bluffs flora (MacGinitie, 1941), is below the lowest part of the section shown in figure 8.

While drilling along the margin of the hydraulic diggings at North Columbia, several biotite-rich tuff beds were found. These beds lie within 25 feet of the ground surface and within 20 feet of the contact between the gravel and the volcanic sediment (pl. 2, hole A). The biotite, generally fresh looking, was dated by potassium-argon analysis at 37.9 ± 1 m.y. (million years). (Analyses were by Lois Schlocker and Jarel von Essen, U.S. Geol. Survey, Menlo Park, Calif.) This date, roughly the Eocene-Oligocene boundary, suggests that the upper half of the upper gold-bearing gravel was deposited during the late Eocene.

Approximately 18 airline miles north of the study

area, in the vicinity of the town of La Porte, a dacite tuff is preserved in one of the channels containing gold-bearing gravel. The tuff was deposited in a depression cut into the upper prevolcanic gravel unit and a coarse volcanic gravel (fig. 15) that may be correlative with the coarse volcanic gravel exposed in the Chalk Bluffs (fig. 8). The dacite tuff at La Porte has been dated by potassium-argon methods at 32.4 m.y. (Evernden and James, 1964). A leaf flora within the tuff is dated as early Oligocene (Wolfe and others, 1961). These dates at La Porte nicely fit the age determined for the rocks in the North Columbia pit and indicate that volcanism began in the general area near the beginning of the Oligocene. The Chalk Bluffs flora further indicates that most of the gold-bearing gravel, including all the lower gravel and some of the upper gravel unit, was deposited prior to the middle Eocene. This period of deposition may have taken all of the early Eocene and perhaps part of the Paleocene.

The Ione Formation of Eocene age (Merriam and Turner, 1937, and Bowen, 1962), exposed sporadically along the east margin of the Great Valley, has been correlated with the gold-bearing quartz gravel (Allen, 1929). The formation has been shown to consist of two members (Pask and Turner, 1952), the lower of which is compositionally more mature than the upper. If there is a correlation between the Ione Formation and the prevolcanic gold-bearing gravel, it would seem that the lower member of the Ione would correlate with the upper member of the gold-bearing gravel. The primary basis for this correlation would be the approximate equivalence of age and similarities of gross lithology, including the heavy-mineral suites. Compositionally immature sedimentary rocks underlying the Ione, informally called lower Eocene deposits of the Foothills by Bateman and Wahrhaftig (1966, p. 129), may then correlate with the lower gold-bearing gravel unit.

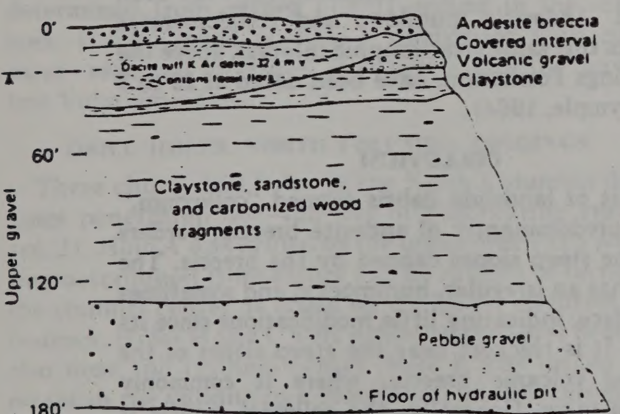


FIGURE 15.—Section of rocks exposed in hydraulic pit near La Porte.

SEDIMENTARY AND VOLCANIC ROCKS

Capping the ridges between the major drainages is a blanket of sedimentary and volcanic rocks of diverse textures. These rocks have been divided into two units for mapping purposes—a lower unit composed of sedimentary rocks rich in volcanic detritus and an upper unit made up of a coarse andesite breccia, andesite cobble gravel, and rhyolitic tuff.

VOLCANIC SEDIMENTARY ROCKS

The unit mapped as volcanic sedimentary rocks comprises coarse gravels, sands, silts, clays, and tuffs, all characterized by a high percentage of volcanic-derived clastic material. The contact with the underlying gold-bearing gravel has been discussed. Sections containing volcanic sedimentary rocks are shown on plate 2 and figures 8 and 15. These rocks occur discontinuously below the overlying andesite breccia. The upper contact with the breccia characteristically is erosional; and it is presumed that, although originally widespread, the rocks were eroded from many areas prior to deposition of the andesite breccia. The only subunit within this map unit that appears to be at all persistent across the region is a coarse cobble gravel, observed at Chalk Bluffs (fig. 8), Scotts Flat, Malakoff Diggings (pl. 2), and La Porte (fig. 15). At Chalk Bluffs and La Porte, it occurs as the lowest subunit. The gravel contains both andesite and nonvolcanic cobbles and pebbles. The volcanic gravel at Scotts Flat, about 20 feet thick, contains gold equal to \$0.20 to \$0.30 per cubic yard, as determined from two large pans of gravel.

Rhyolitic tuffs, both fresh and reworked, are present at Chalk Bluffs and below the Columbia Hill lookout on San Juan Ridge, north of Nevada City. Chalk Bluffs derived their name from a massive white tuff that forms a prominent cliff bordering the hydraulic pits near Red Dog and You Bet. A thin tuff bed overlying the gravel near North Columbia contains the biotite dated at 37.9 ± 1 m.y.

The volcanic sedimentary rocks reach a maximum thickness of 120 feet at Chalk Bluffs and along the Graniteville Road below the Columbia Hill Lookout.

Part of this section may correlate with the Valley Springs Formation (Piper and others, 1939), which is composed of rhyolite tuffs and associated clastic rocks, but the date of 37.9 m.y. for the tuff at North Columbia suggests that these rocks may be somewhat older than the Valley Springs, which was dated as 20–30 m.y. (Dalrymple, 1964).

ANDESITE BRECCIA AND GRAVEL

The upper unit is composed primarily of andesite breccia with intercalated cobble and boulder gravel. The breccia is poorly sorted and fragments are angular; some blocks are as much as 6 feet in diameter; commonly the fragments are 6 inches to a foot in diameter.

The moderately well indurated light-gray matrix is composed of fine sand-size andesitic debris. At several localities, zones containing rounded, apparently water-worn cobbles and boulders are intercalated with the breccia. A typical gravel horizon is present within the breccia on San Juan Ridge exposed along the road leading down to Bloody Run Creek (fig. 16). Several such horizons were encountered 400–500 feet below the top of the breccia on San Juan Ridge. These boulder zones are good aquifers, and springs are common where they crop out, particularly in a moderately thick section of andesite gravel along the Relief Hill Road above the old mining town of Relief.

Thick deposits of andesite gravel, formerly believed to belong to the older volcanic unit, cap Montezuma Hill, Bunker Hill, and Round Mountain. The exposures at Round Mountain show stratigraphic continuity with the andesite breccia and are mapped as part of the younger volcanic unit.

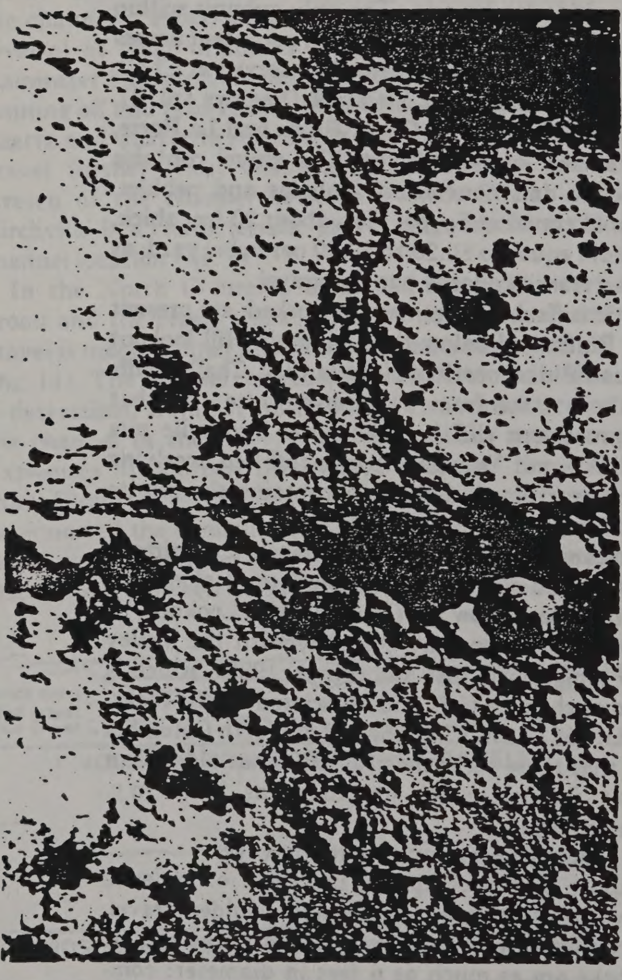


FIGURE 16 —Exposure of andesite breccia overlying andesite boulder gravel on San Juan Ridge along road to Bloody Run Creek. Largest boulder in photo is 3 feet in length.

The four drill holes on San Juan Ridge penetrated 510–550 feet of volcanic breccia and intercalated gravel. The thickness varies markedly on Washington and Harmony Ridges, where the breccia was deposited across the boundary of a highland surface extending to the east and a somewhat flatter, lower surface occupied by the ancestral Yuba River to the west.

The breccia seems to be made up of a series of mudflows separated by a few preserved soil horizons. The flows eventually filled most of the depressions in the land surface, forming a broad, fairly flat surface. Remnants of this constructional surface are preserved as the west-sloping interfluves of the northern Sierra. The concordant elevation and slope of this surface across the drainages is easily mistaken for an old erosion surface.

A rhyolitic, biotite-rich tuff is present in the southern part of the area along the Southern Pacific Railroad tracks between Alta and Dutch Flat. Samples of the tuff were collected from a roadcut along the paved highway leading to the California Division of Forestry Headquarters near the town of Alta, where the tuff overlies both bedrock and a thin section of cobble gravel. A potassium-argon date on biotite from the tuff was determined at 8.7 ± 0.5 m.y.; this date indicates that the tuff correlates with latites that are farther south in the Sierra Nevada which are dated 8.8 to 9.0 m.y. (Dalrymple, 1963).

This upper unit of andesite breccia and gravel resembles the Miocene and Pliocene Mehrten Formation described from other areas in the Sierra Nevada (Piper and others, 1939; Curtis, 1954), as well as the Miocene and Pliocene Disaster Peak and Relief Peak Formations of Slemmons (1966) of the central Sierra Nevada. Direct correlation with these formations has not been established. Andesitic mudflows and conglomerates deposited in this same stratigraphic interval, above the Valley Springs Formation, have been dated at 19 to 5 m.y. (Dalrymple, 1964).

COLLUVIUM

A blanket of landslide debris termed "colluvium," composed predominantly of andesite breccia, borders many of the steep slopes capped by the breccia. The colluvium has an irregular, hummocky, and sometimes blocky surface, indicating little modifications since its formation. It is thickest near the steep slopes of the undisturbed volcanic breccia, where it commonly grades into talus. Generally, the colluvial mantle is several tens of feet thick and may extend as much as 2 miles from the volcanic cliffs.

The colluvium is perhaps thickest and most wide-

spread at the locations where the prevolcanic-gravel-filled channels disappear beneath the volcanic rocks capping the ridges. The weak clay in the upper unit of the prevolcanic gravel is probably instrumental in the common slope failure at this location. Slope failure might be expected where a thick layer of relatively porous, water-saturated volcanic rocks overlies weak clays. Consequently, the presence of colluvium can be used as an indicator of buried gold-bearing gravel that might otherwise be unsuspected.

Colluvium undoubtedly accumulated as soon as erosion began to remove the volcanic breccia, and as the breccia scarp retreated, a mantle of colluvium continued to form at its base. The rate of erosion and colluvium accumulation probably reached a maximum during the wet periods of the Pleistocene, and most of the present colluvium probably formed at that time.

PLACER GOLD

Any study of the characteristics and distribution of gold in the Tertiary gravel must be undertaken with the realization that the most accessible and readily attained gold has been removed by the mining methods of the late 19th and early 20th centuries. Information on gold values of the mined-out deposits can be obtained only from scanty information recorded by the early miners, some mining companies, and an occasional geologic report. Only in those areas where the gold-rich portion of the gravel is covered by thick deposits of low-grade gravel, volcanic breccia, and colluvium is it now possible to obtain first-hand information about the occurrence of the Tertiary placer gold.

One of the major objectives in drilling three holes in the thick gravel in the North Columbia diggings was to obtain samples of a complete unmined gravel section in order to study the gold characteristics and vertical distribution and to compare values with gold values determined from earlier (1939) drilling in the same area. It also made possible a comparison with gold values reported for gravel elsewhere within the ancestral Yuba drainage.

DRILL HOLES, NORTH COLUMBIA DIGGINGS

Three churn drill holes in the North Columbia diggings penetrated 308–455 feet of gold-bearing gravel (pl. 2). Hole A was drilled on the upper bench at a position determined from earlier (1939) drilling to be near the channel center as based on the lowest elevation of bedrock. Holes B and C were drilled on the lower bench, also near the channel center. Hole B was drilled to establish the validity of a geophysical interpretation of depth to bedrock—an interpretation that did not fit the information derived from the earlier drilling. Hole B also provided data useful in comparing gold values over a short interval (<200 feet) laterally across the channel. All three holes were drilled 10 feet from earlier drill holes, a distance that permits comparison of gold values without the hazard of encountering cave-in material and disturbed ground associated with the uncased holes.

All the holes were drilled with a cable-tool drill, also termed a churn drill, placer drill, or keystone drill, using a 6-inch drive pipe and a 7.5-inch drive shoe. Casing was driven to a depth of 398 feet in hole A, 232 feet in hole B, and 226 feet in hole C, at which depths the drill encountered boulders or cemented gravel sufficiently hard to prevent further driving. The gravel was well cemented below the bottom of the casing, allowing an open hole to be drilled without fear of caving. All the material drilled in all three holes was collected, and the samples separated every 5 feet in the upper, gold-poor gravel and every 1–3 feet in the lower high-value gravel. Drilling of hole A was begun September 4, 1968, and hole C was completed December 9, 1968.

METHODS OF GOLD RECOVERY AND VALUE DETERMINATION

Samples were panned at the drill sites as they were "bailed" from the drill hole (fig. 17). Gold colors were noted from each panned concentrate and the concentrate dried and stored in glass vials. As a check to determine the effectiveness of recovering fine gold from the drill hole, eight distinctive gold flakes ranging in diameter from 2 mm to 0.25 mm were dropped and washed down the hole when drilling at a depth of about 150 feet within the gold-poor section. It was reassuring to recover all eight gold flakes within the next two samples collected, following each drive of 5 feet.

In the laboratory, gold values were determined for each panned concentrate by handpicking and weighing the detrital gold fragments. Amalgamation with mercury was not done for several reasons. It seemed important not to destroy the original size and surface characteristics of the gold fragments useful for further study, and it was discovered that as much as 20 percent of the gold in some of the lower samples would not amalgamate. "Roughing up" the gold in the pan, cleaning with the ultrasonic probe, and boiling in separate acid solutions of HCL and HNO_3, although improving the amalgamating properties, did not allow recovery of all the gold. Using a binocular microscope and a large hand magnifying glass, it was possible to effect a clean separation rather quickly. As most of the gold fragments were larger than 1 mm in diameter, the grains were easily seen and quickly removed with the aid of a fine camel's-hair brush. The gold from each sample was weighed to the nearest tenth of a milligram. Knowing the diameter of the drill hole and the length of hole represented by each sample, it was a simple matter to

FIGURE 17.—Panning churn drill samples from hole A in the hydraulic pit near North Columbia. Panner, August Ebbert, is a former hydraulic miner.

calculate gold values. The value of gold in cents per milligram was determined using a price of $35.00 per ounce and a fineness of 885, the average fineness of placer gold obtained in California (Wells, 1969, p. 99). The following formula was used to determine gold values, V, in cents per milligram

$$V = \frac{\$\text{Price} \times \text{fineness} \times 0.1}{31{,}103} \text{ (Wells, 1969, p. 121),}$$

substituting values

$$V = \frac{35 \times 885 \times 0.1}{31{,}103} = 0.0996 \text{ c per mg.}$$

This figure was sufficiently close to 0.10 cent to round off for subsequent value calculations.

The "Radford" factor (Wells, 1969, p. 35) was used in computing hole volume. Theoretically a 7.5-inch drive shoe should cut 0.3068 square feet of material, but because of rounding and abrasion, it probably cuts less. The "Radford" factor implies a 7.5-inch drive shoe will cut, on an average, 0.27 square feet of material. A 1-foot drive would include a sample volume of 0.27 cubic feet or 1/100 cubic yard. Using the "Radford" factor, gold values can be computed by the following formula

$$\text{cents per cubic yard} = \frac{W \times V \times 27}{A \times D} \text{ (Wells, 1969, p. 34),}$$

where

W = weight of gold in milligrams,
V = value of gold in cents per milligram,
A = effective area of drive shoe in square feet,
D = length of sample in feet, and
27 = cubic feet in cubic yard.

For example, value calculations for a 2-foot section of the lower part of hole A (422–424 ft.), if

W = 45 milligrams of gold,
V = 0.1 (as just calculated),
A = 0.27 (Radford factor), and
D = 2,

$$\text{cents per cubic yard} = \frac{45 \times 0.1 \times 27}{0.27 \times 2} = \$2.25.$$

GOLD VALUES OLD AND NEW

Gold values in dollars per cubic yard are shown for holes A, B, and C on plate 2. In all three holes, the lower 80–100 feet contains the bulk of the gold. The richest sample consisted of a 2-foot interval in hole B (277–279 feet), which contained gold equal to a $6.35 per cubic yard. A 5-foot section of hole C (289–294 feet) gave a value of $2.80 per cubic yard, the highest for that hole; and the interval 420–422 feet in hole A produced values of $2.60 per cubic yard, the highest for hole A. Above the high-value zone in the lower part of each hole, values do not exceed 12 cents per cubic yard and rarely exceed 2 cents per cubic yard.

The upper 10 feet of bedrock in holes B and C contain appreciable gold, probably gold lodged in the fractures of the bedrock by fluvial processes and not lode gold. The values drop off rapidly as the holes penetrate deeper, and presumably fresher, less-weathered bedrock. The old adage that gold values are always highest on the bedrock does not seem to obtain here. In hole A gold values are quite low on the bedrock and increase upward to a maximum at 12–25 feet above bedrock. Clay beds in the lower gravel, if correctly placed as interpreted from drilling characteristics, do not seem to act as barriers or false bedrock horizons, for gold values do not seem to be in any way related to them.

Although holes B and C are separated by no more than 200 feet horizontally, there is considerable variation in gold content between the two. Other than the lower bedrock contact, it would be difficult to correlate any overlying zones solely on the basis of gold values. However, the 1939 drilling indicates high values near the 250–260-foot level (fig. 18) for all three holes.

Samples from the 1939 drilling yielded considerably higher values than samples from the 1968 drilling for holes at the same locations (fig. 18). Hole B was drilled to a depth of only 145 feet in 1939, at which point the clay-rich material penetrated was interpreted to be weathered bedrock. A refraction seismic profile that was run across the location of hole B in 1967 indicated deeper bedrock, and the 1968 drilling program proved this interpretation to be correct. Real differences in gold values due to the slight shift in position of the drill holes would be expected to balance. Weighted averages show that the differences do not balance; rather, values obtained by the 1968 drilling are approximately 50 per-

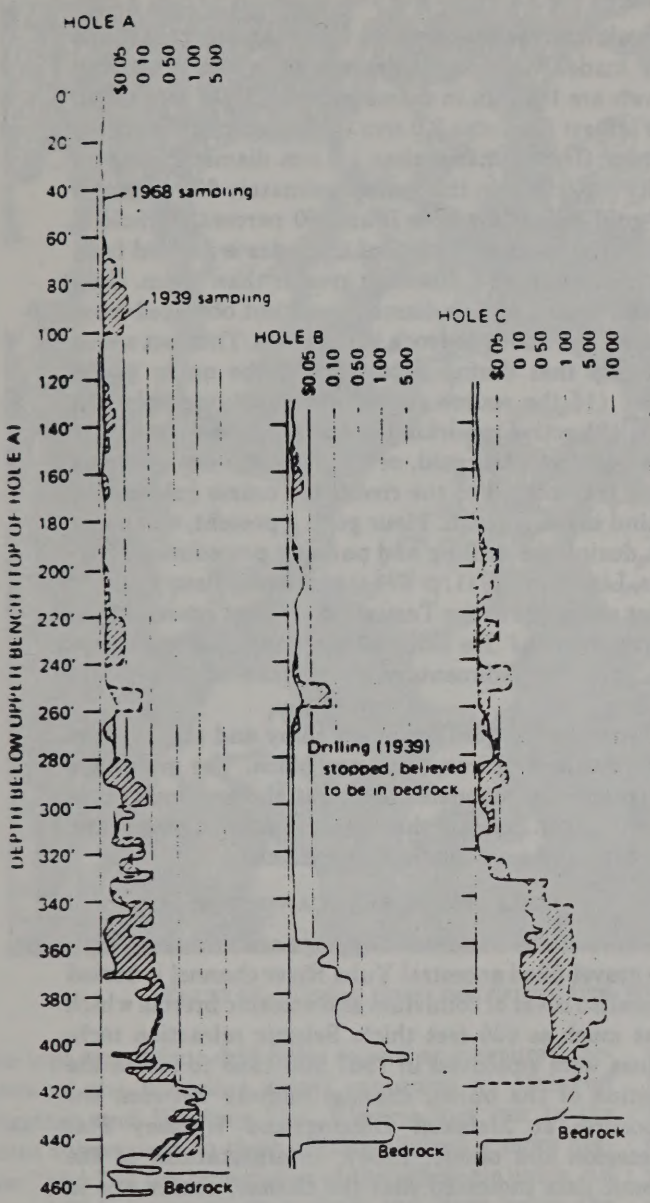


FIGURE 18.—Comparison of gold values (per cu yd) obtained by drilling, 1939 and 1968, North Columbia hydraulic pit.

cent of those obtained from the 1939 drilling (table 1). Different methods of recovery, sample concentration, and value calculation must be responsible for the consistent difference in evaluation. It is impossible to say with the information available which evaluation is more correct.

A total of 61 holes were drilled and sampled in the North Columbia diggings in 1939. Using the drill records from these holes (made available by the property owners), it was determined that 21.8 million cubic yards of the lower gravel contained $17.7 million in gold at an average grade of 81.2 cents per cubic yard (Peterson and others, 1968, p. 10). To conservatively evaluate the gold-producing gravel, we should probably consider reducing the total figure obtained from the 1939 drilling by as much as 50 percent (table 1). This gives a total of $8.6 million. It must be kept in mind that this figure is more correct only if the gold values obtained from our 1968 sampling program are more accurate than those obtained in 1939.

Gold values for upper and lower gravel at selected localities obtained from both published and unpublished reports on the production records and drilling (sampling) in the area are given in table 2. Some caution is advisable in evaluating the data, for much of this information, before being permanently recorded, existed as hearsay or speculation passed by word of mouth. If, however, it is assumed that these figures are approximately correct, it is apparent that the values vary widely from place to place along the channel. The relatively high values at Liberty Hill, French Corral, and Smartville probably can be attributed to the almost exclusive presence of the lower gravel at those places. The thick low-grade upper gravel prevalent in many of the other areas was virtually absent at these localities.

In figuring the averages, the abnormally high values of the lower gravel at Derbec and in the area from Dutch Flat to Indiana Hill were excluded because these values were determined from the unusually rich gravel mined within only a few feet of bedrock. The averages of $0.13 and $0.59 per cubic yard for the upper and lower gravel, respectively, probably are representative values in generalizing about the vast amount of gravel along the major tributaries of the ancestral Yuba River.

SIZE, SHAPE, AND SURFACE CHARACTERISTICS OF THE GOLD

The unamalgamated detrital gold recovered from drill holes A, B, and C was studied with the aid of a

TABLE 1.—*Comparison of gold values obtained from drilling, 1939 and 1968, in North Columbia hydraulic pit at approximately the same location*

[Values expressed as dollars per cubic yard at gold prices of $35 per oz]

Drill hole		Sections	Depth (ft)	Dollar value 1939	Dollar value 1968
A		Upper	375	0.036	0.016
		Lower	80	.62	.49
		Entire	455	.14	.10
B		Upper	230	.017	.007
		Lower	78	......	.81
		Entire	308	.017	.21
C		Upper	210	.031	.013
		Lower	100	2.43	.668
		Entire	310	.805	.223
	Average	Upper		.028	.012
		Lower		1.53	.66
		Entire		.32	.17

TABLE 2.—*Gold values in dollars, obtained from mining records and drill sampling for selected areas along the ancestral Yuba River*

[Values adjusted to a $35.00 per oz gold price]

Location	Upper gravel	Lower gravel	Undifferentiated	Source
Gold Run	0.08	0.30	0.24–0.32	Hydraulic Mining Comm. of Calif.
Dutch Flat to Indiana Hill	.19	(up to 15.00)	......	Lindgren, 1911, p. 71.
Dutch Flat	.185	.68	.43	Hydraulic Mining Comm. of Calif.
Liberty Hill	.24	1.40	.24–.39	Do.
Christmas Hill, Little York Diggings	.115–.17	.34	.27	Do.
You Bet, Red Dog	......	......	.15	Do.
Hunts Hill, Quaker Hill, Buckeye Hill	.11	......	.255	Do.
Scotts Flat	.10	......	......	Lindgren, 1911, p. 43.
Sailor Flat, Blue Tent	.034	......	.12–.24	Hydraulic Mining Comm. of Calif.
Blue Tent	.042	.255	......	Lindgren, 1911, p. 71.
Omega	......	......	.23	Do.
Relief Hill	......	......	.15–.22	Hydraulic Mining Comm. of Calif.
Snow Point	......	......	.19	Do.
Orleans Flat, Woolsey Flat	......	......	.254	Do.
Derbec	......	6.30 (lowest gravel)	......	Lindgren, 1911, p. 71.
Malakoff Diggings	.049–.067	.56	......	Do.
Malakoff Diggings	.065	.56	.165	Hydraulic Mining Comm. of Calif.
Malakoff Diggings to Badger Hill Diggings	......	......	.15–.20	Do.
North Columbia to Cherokee	.17–.25	......	......	Lindgren, 1911, p. 71
North Columbia to Badger Hill Diggings	.18	.39	.20	Do.
Badger Hill Diggings	.06	.26	......	Unpub. mine records.
North San Juan to Smartville	......	......	.51–.63	Lindgren, 1911, p. 71.
French Corral	.24	.70	.50	Unpub. mine records.
Smartville	......	1.05	......	Do.
Averages	.13	.59	.27	

binocular microscope, and the following generalizations were made. Most of the fragments in the lower rich gravels are 1–2 mm in diameter and 0.1–0.2 mm thick. The largest flake was 3.0 mm in diameter, the smallest 0.3 mm. Grains smaller than 1.0 mm diameter are generally 0.1–0.05 mm thick. Approximately 80 percent of the gold weight for hole B and 50 percent for hole C within the lower rich parts of the holes is derived from gold fragments of a diameter greater than 1 mm. Gold coarser than 1 mm in diameter was not observed more than 80 feet above bedrock in the holes. This fact seems to imply that during deposition of the upper gravel either (1) the source region was supplying only fine gold; (2) active reworking in the rivers was physically breaking down the gold, or (3) only the fine gold was being transported to the rivers, the coarse gold lagged behind in the regolith. Flour gold, if present, was likely lost during the drilling and panning procedures. However, Lindgren (1911, p. 67) states that, "flour gold *** is not abundant in the Tertiary or present gravels of the Sierra Nevada." He believed the flour gold was swept out into the sedimentary rocks that fill the Great Valley.

Generally the gold flakes are shiny and rough; some show coatings of iron oxide and silica. The grains are flattened with rounded edges that show general abrasion (fig. 19). Some grains have particles of magnetite, ilmenite, and quartz adhering to them.

DRILL HOLES, SAN JUAN RIDGE

Between the Malakoff Diggings and Woolsey Flat, the gravel-filled ancestral Yuba River channel is buried beneath a cover of colluvium and volcanic breccia which is as much as 650 feet thick. Seismic refraction techniques were employed in 1967 and 1968 to locate the position of the buried channel midway between the exposures at Malakoff Diggings and Woolsey Flat (Peterson and others, 1968). Interpretations of the seismic data indicated that the channel center was in the vicinity of the Upper Derbec Spring along the Graniteville Road and was covered by approximately 900 feet of volcanic breccia. A drilling program was set up to drill through the thick volcanic breccia and the suspected underlying gravel to bedrock. During the late summer and fall of 1968, four holes were drilled (rotary drill) along the Graniteville Road in an attempt to prove the geophysical interpretation of the channel center location and to sample the suspected rich lower gravel. "Ditch" samples separated from the mud by a shaker with a 30-mesh screen were collected every 5–25 feet for the purpose of noting lithologic characteristics and concentrating heavy minerals, gold particularly. Silt and clay-sized cuttings that filtered below the shaker were also collected and examined. Plate 2 shows

FIGURE 19.—Typical placer gold derived from lower gravel in North Columbia hydraulic pit.

the logs for all four drill holes as well as measured sections of the bounding gravel exposures at Malakoff Diggings and Woolsey Flat. Electric logs run in the holes subsequent to their drilling allowed a more precise "pick" of the boundaries of the lithologic units. All four drill holes penetrated sands and clays below the volcanic breccia and above the bedrock. Thickness of the volcanic breccia ranged from 500 to 650 feet. The samples collected from the gravel section were panned by hand and examined for gold. As gold was absent in most samples and rare in others, it was not worthwhile to construct plots of gold content similar to those produced for holes A, B, and C in the North Columbia diggings. Values determined were less than $0.01 per cubic yard. The heavy-mineral plots are discussed under the section "Other Heavy Minerals." The disappointingly low gold values, together with the fine-grained textural characteristics of the section, clearly suggest that the part of the gravel drilled through correlates with the upper sections as exposed at Malakoff Diggings and Woolsey Flat (pl. 2). It seems that all four holes were drilled parallel to and along the margin or flood plain of the channel as it passes beneath the volcanic rocks on San Juan Ridge. In the light of the information obtained from the drilling, the suspected trace of the buried channel center was redrawn (fig. 2).

GOLD SOURCE

Evidence suggests that the bulk of the gold in the Tertiary gravel has been derived from the gold-bearing quartz veins occurring within the low-grade metamorphic rocks of the Sierras. The gravel that has the highest gold values also contains abundant detrital fragments of white "vein" quartz and blue-gray siliceous phyllite and slate common throughout the foothill region. Furthermore, high concentration of gold in the gravel is essentially restricted to the areas where the ancient rivers crossed the gold-bearing quartz veins in

the bedrock of the foothills region (Lindgren, 1911, p. –66).

In order to compare the composition of the placer gold with the lode gold of the Sierra Nevada, trace-element contents were determined for 20 samples of tive gold (table 3). Nineteen of the samples were ected from detrital gold flakes recovered from drill holes A and C within the North Columbia hydraulic pit. e analyses of these samples can be compared with e analyses of a small nugget from the bedrock lode mines of the Alleghany district, a likely source for the gold in the gravel at North Columbia. Except for the lver common in nearly all samples of native gold, only pper is present in all samples; the copper content ranges from 100–1,000 parts per million. Small amounts lead and iron are present in some samples. These ata show that the placer gold is indeed similar to the lode gold indigenous to this part of the Sierra Nevada and suggest that the gold was derived locally.

The bedrock gold mines in the northern Sierra evada included in this study show a distinct spatial relation to outcrops of ultrabasic rocks (fig. 2). Most f the quartz veins and mines are either adjacent to the ltramafic rock bodies or within several miles to the east of them, a relation recognized by the early miners and reported in more recent studies (Ferguson and annett, 1932). These areas probably provided a rich ource for much of the gold within the ancestral Yuba River drainage. Of course, much of the gold in the gravel was derived from higher level rocks that have been completely removed by erosion. The principal gold-producing quartz veins in the Alleghany district dip 25°–40° north to northeast (Ferguson and Gannett, 1932, p. 29) and project at higher eroded levels several miles to the south and southwest. These veins may have provided the bulk of the gold produced from the gravel at Moore's, Orleans, and Woolsey Flats, and the Derbec and Malakoff Diggings areas; they may have provided a significant portion of the gold in the hydraulic diggings farther down the channel as far as Smartville. The vein system in the Nevada City–Grass Valley area probably contributed a significant portion of the gold recovered from the gravel in that area as well as in the Blue Tent and Sailor Flat diggings. Rich gravels in the You Bet to Dutch Flat area may have been derived from a source south of the mapped area as well as from northeast of Dutch Flat.

TABLE 3.—*Trace-element content of native gold from placers and lode deposits, San Juan Ridge, Calif.*

[Analyst: A. L. Sutton; analyses by standard six-step spectrographic analysis. Elements looked for but not detected: B, Ba, Be, Bi, Cd, Co, Ga, La, Mn, Mo, Pd, Pt, Sc, Sn, Ti, V, Y, Yb, Zn, Ir, Os, Rh, Ru. N, not detected, at limit of detection or at value shown; L, detected, but below limit of determination or below value shown]

Sample No.	Weight (mg)	Fe (percent)	Ag (ppm)	Cr (ppm)	Cu (ppm)	Pb (ppm)
Detrital gold from drill holes in North Columbia hydraulic pit						
A126–128	5.88	L.003	120.000	N	300	N
A138–142	9.85	.005	270.000	N	200	N
A142–145	6.01	N	92.000	N	300	N
A157	5.60	N	31.000	N	100	N
A158	6.62	N	71.000	N	100	N
A159	6.48	.007	73.000	20	100	N
A160	6.87	L.033	64.000	N	150	L
A161	6.95	L.003	86.000	N	150	N
A162–164	9.75	L.002	92.000	N	100	N
C75–77	4.60	L.005	62.000	N	1.000	N
C92–96	7.34	.003	56.000	N	500	N
C97–101	9.96	L.002	90.000	N	500	N
C104	9.80	N	59.000	N	300	L
C105	7.60	N	94.000	N	300	L
C106	9.38	L.002	64.000	N	1.000	L
C107–108	6.56	L.002	52.000	N	1.000	L
C109	7.00	L.002	76.000	N	300	20
C110–112	7.69	L.002	64.000	N	500	L
C101A–109A	6.86	L.002	130.000	N	200	L
Lode gold from Alleghany district						
W Y -S26-69	5.37	L.002	170.000	N	150	L

GOLD OCCURRENCE

Why does the bulk of the placer gold occur in the lower 80–100 feet of gravel? It is worthwhile to examine several possible explanations.

It is presumed that a lower, relatively thin layer of compositionally and texturally immature gravel was always present on the river channel floor during the long interval of downcutting. This thin veneer of gravel represented the coarse material being transported by the river much as does the gravel in the bottom of the Colorado River today (Berkey, 1935). While most of the constituents of the gravel were undergoing lateral transport along the river during times of flooding and high runoff, the gold that was constantly being added to the gravels was lagging behind, much as it does in a sluice box. As the volume of gravel in the river bed essentially remained the same through additions of new detritus and removal of old, the gold concentrations gradually increased. Most of the gold that was eroded during the early downcutting history of the ancestral Yuba River ended up within this giant sluice box. Certainly some gold was flushed beyond the river and wound up in the marine sediments to the west. As the river ceased to downcut and began aggrading, the nature of the weathering and erosion of the surrounding hill slopes changed. Whereas physical weathering was predominant in breaking up and moving the coarse detritus on the steep side slopes, chemical weathering became common as the slopes became less steep and thick soils were able to develop. Only the most resistant minerals survived the intense weathering process. Judged by the amount of vein-quartz pebbles in the upper gravel, a great deal of potential gold-rich bedrock was eroded. What happened to the gold? Certainly the low gold content of the upper gravel does not reflect all the gold that was ultimately released. Several possi-

bilities come to mind: (1) the gold was physically and chemically broken down to a size small enough to be transported by the river and was deposited offshore in the marine environment; (2) the prevailing conditions of chemical weathering facilitated the removal of the gold in solution; or (3) the gold became concentrated in the soil zone and, because of the low slope gradients, was not transported in high concentrations into the rivers. If indeed the gold was not moved into the rivers (3), it should still be present where the old soil zones are preserved, primarily beneath the volcanic cover. Perhaps much of the gold that became concentrated in the younger rivers (intervolcanic and recent) was derived from this old gold-rich regolith.

A more definitive solution to this problem will have to await (1) exploration for gold in the upper Eocene sediments in the Great Valley and (2) laboratory study of physical and chemical resistance of gold in a variety of environmental conditions.

GOLD RESOURCES

Since the days of the famous Sawyer decision in 1884, when most of the hydraulic mining ceased, people have calculated volumes and values of minable gravel remaining along the Tertiary river courses. As the costs of mining, water, and gold changed, these figures were reevaluated, recalculated, and somewhat refined. It has long been known that a sizable gold resource remains dispersed in the gravel. Whereas equipment and labor costs markedly increase oweing to inflation, the price of gold remains virtually the same, thereby greatly reducing the incentive to mine this tremendous resource. With the presently known and allowable mining techniques, it is understandable that in recent years few mining ventures have been attempted and that only a small percentage of these have survived.

Lindgren (1911, p. 81) estimated that approximately \$507 million (\$35 per oz) in gold was produced from the Tertiary gravel. Estimates of the gravel yardage removed in deriving this \$507 million in gold was subsequently calculated at 1,585 million cubic yards (Gilbert, 1917), an average yield for the entire body of mined gravel gives \$0.32 per cubic yard. This figure is probably too high a value to use in computing total gold content of the remaining gravel, as much of the unmined gravel is the gold-poor upper gravel.

Probably the most detailed and complete determinations of volume of unmined gravel were made by Arthur Jarman (1927) in his report of the Hydraulic Mining Commission to the California State Legislature. This report is on the feasibility of resuming hydraulic mining by construction of debris dams. Table 4 draws on much of the information contained in Jarman's report. Figure 20 shows where gold-rich lower gravel is believed to lie along the ancestral Yuba River channel.

The \$0.193 per cubic yard average obtained by dividing the estimated total gold value of \$188,085,000 by the total volume of unmined gravel, 977,440,000 cubic yards. This figure is substantially lower than the \$0.32 per cubic yard value obtained by using Lindgren's (1911) and Gilbert's figures (1917) and presumably reflects the lower tenor of the remaining gravel.

As can be seen from table 4, the gravel and gold resource between Malakoff and Badger Hill Diggings accounts for more than 75 percent of the total from all areas considered. The figure of \$0.175 per cubic yard (from estimates by Jarman (1927) on the Hydraulic Commission of California), which is used to obtain the \$140 million total gold value, is similar to the average value of \$0.17 per cubic yard obtained from the 1968 drilling in the North Columbia diggings (table 1). The mapping for this study did not include all the Tertiary gold-bearing gravels known to exist throughout the foothill region of the Sierra Nevada. From field reconnaissance checks and a study of published and unpublished accounts of the unmined gravel, however, it is doubtful that an area exists in which gravel quantities and gold values exceed those in the Badger Hill–Malakoff region. It would seem, therefore, that the deposit between Badger Hill and Malakoff would be the most likely target for a resumption of large-scale mining of the Tertiary gravel.

The figures for unmined gravel in table 4 do not include deposits beneath the volcanic cover on San Juan, Harmony, or Washington Ridges. Our knowledge of the occurrence and gold values of gravel beneath the volcanic rocks is meager at best, but it is doubtful

TABLE 4.—*Areas containing appreciable unmined gravel with estimated gold content along the ancestral Yuba River drainage*

[Gravel volumes from Jarman (1927) gold values (per cubic yard) from table 2]

Area	Volume of unmined gravel (cubic yards)	Estimated average gold value per cubic yard (\$35 per oz)	Estimated total gold content (\$35 per oz)
Gold Run	75,000,000	\$0.28	\$21,000,000
Liberty Hill	4,500,000	.315	1,400,000
Dutch Flat	28,500,000	.43	12,000,000
Christmas Hill, Little York Diggings	3,440,000	.27	930,000
You Bet, Red Dog	20,000,000	.255	5,100,000
Hunts Hill, Quaker Hill, Buckeye Hill	5,000,000	.255	1,275,000
Omega	24,000,000	.225	3,240,000
Relief Hill	15,000,000	.185	2,780,000
Sailor Flat, Blue Tent	2,000,000	.18	360,000
Malakoff Diggings to Badger Hill Diggings	800,000,000	.175	140,000,000
Totals	977,440,000	193	188,085,000

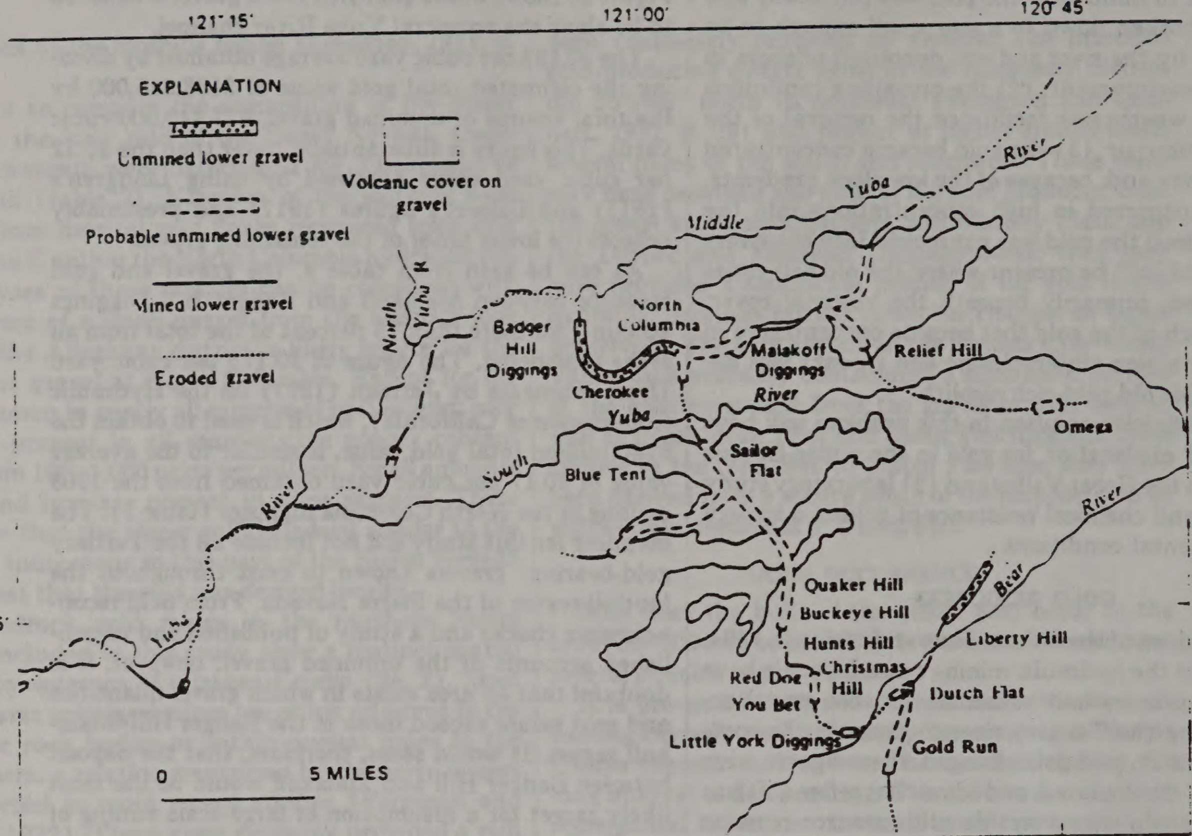


FIGURE 20.—Part of the ancestral Yuba River drainage showing areas of mined and unmined relatively rich gravel along the river channel.

that the total volume of material would increase the calculated totals by as much as 50 percent.

No method has yet been devised for recovering the gold from the Tertiary gravel more efficiently than the hydraulic. In referring to the rich deposit on San Juan Ridge, Jarman (1927) wrote, "It does not appear to be possible to work this gravel profitably by any other method than that of hydraulicking," and that statement seems appropriate today. Even "in place" leaching, now receiving some consideration by various groups, was discarded by Jarman as a poor method of removing the gold from the gravel. The economic recovery of the gold dispersed through the widespread Tertiary gravel remains a challenge to the keenest minds in the mining industry. It is not unreasonable to foresee the day when the gold will be removed as a byproduct of the gravel used principally for construction and decorative uses. Current prices (1970) in the San Francisco Bay area for decorative rock of the type most common throughout the "upper" Tertiary gravel is $19-$28 per cubic yard!

SECONDARY SULFIDES

Common throughout much of the lower "blue gravel" are sulfides, primarily pyrite. They occur within the sandy matrix, form coatings on pebbles and cobbles, and are most plentiful in areas marked by an abundance of carbonized wood. The fresh and shiny crystals are quite clearly of a secondary nature, having formed after deposition of the gravel. The pyrite accumulations are restricted to gravel zones below the water table. A deep-red color of the overlying gravel is produced by oxidation of the contained pyrite as a result of lowering of the water table and exposure to the air. Pyrite accounts for more than 50 percent of the heavy-mineral fraction within portions of the lower and middle exposures of gravel in the North Columbia pit (pl. 2). X-ray diffraction patterns of heavy-mineral concentrates derived from drill cuttings in the North Columbia diggings reveal that all the sulfide is pyrite and not marcasite or arsenopyrite.

One small mining venture in the vicinity of Birchville in 1967 reported that lower blue gravel was yield-

ing 10 pounds of sulfides per cubic yard of gravel processed. This concentrate, however, was far from being pure sulfide; it contained other heavy minerals.

Nodular and granular pyrite from a gravel mine in Butte County yielded gold at the rate of $173 to the ton (Lindgren, 1911, p. 76). At $20 to the ounce this would be equal to 8.65 ounces of gold per ton of sulfide or approximately 270 ppm. This figure seems abnormally high when compared with atomic absorption gold analyses of pyrite obtained during this study. Seven separate samples of pyrite all showed less than 0.10 ppm gold.

The pyrite within the gravel is most often closely associated with carbonized wood. It exists as nodules, coatings, and thin seams as much as ⅛ inch thick. The genesis of the pyrite is probably similar to that occurring within coal deposits (Newhouse, 1927). An ideal environment for the production of low-temperature pyrite was probably created during and following deposition of the gravels. The prevailing subtropical climate (MacGinitie, 1941, p. 73) very likely promoted development of thick, lush vegetation that was periodically carried into the major drainages where it formed temporary local dams along the channel. Sediment would fill in behind such "log jams" and eventually cover them, thereby preventing their destruction by oxidation. A moderately warm mean annual temperature of 65°F (MacGinitie, 1941, p. 78) may have promoted the growth of anaerobic bacteria active in producing hydrogen sulfide from the buried trees and shrubs. Hydrogen sulfide would have readily reacted with the iron-rich ground water producing FeS that was subsequently changed to FeS_2 (Edwards and Baker, 1951; Galliher, 1933). Eventually, as the water table dropped, much of the pyrite in the upper and middle sections of the coarse gravel layers was oxidized and leached. Siderite and limonite-rich cemented zones a few inches to several feet thick, common throughout much of the middle section of the gravel, probably record the water table positions at various times in the postdepositional history of the gravel.

OTHER HEAVY MINERALS

Other heavy minerals in the panned concentrates were studied to better understand the depositional environment of detrital gold.

METHODS OF CONCENTRATION AND IDENTIFICATION

Bromoform (sp gr 2.85) was used to separate the few light minerals not removed in the hand-panning process. A small magnet was useful in removing drill steel from the samples, and with it the magnetite abundant in a few samples. A relatively nonmagnetic heavy mineral fraction ranging in size from 0.50 to 0.0625 mm (medium sand to coarse silt) was thus produced for study.

Minerals were identified by optical methods and checked occasionally by X-ray diffraction techniques. Because percentages of minerals were estimated, they are not to be regarded as precise figures. Samples selected for examination generally were taken at intervals of 10–15 feet in the churn drill holes, and closer intervals where significant mineral changes were occurring. More widely spaced samples were selected for examination from the rotary drill holes on San Juan Ridge where heavy-mineral contents were relatively uniform.

CHURN DRILL SAMPLES, NORTH COLUMBIA DIGGINGS

Vertical distribution of heavy minerals is plotted along the section for drill holes A and C in the North Columbia diggings (pl. 2). The heavy minerals plot of hole A is based on examination of 41 samples, that for hole C on 20 samples. Pyrite, ilmenite, zircon, siderite, amphibole, epidote, and chlorite are present in quantities large enough (>5 percent) to plot. The category "other" includes rock fragments, sphene, garnet, tourmaline, zoisite, rutile, leucoxene, and pyroxene.

Pyrite, of secondary origin, is present in significant quantities. It seems quite likely that the first appearance of pyrite marks the water table or boundary between oxidation (red gravel) and reduction (blue gravel). This horizon is at a similar level, 2,675 to 2,685 feet elevation, in all three holes, as would be expected if it represents a water table. Ilmenite and zircon make up most of the heavy minerals in the upper (oxidized) part of the gravel (fig. 21). This might be expected because ilmenite is more stable under oxidizing than reducing conditions, and zircon is a very durable mineral both physically and chemically. The lower gravel is characterized by a variety of both stable and unstable mineral types including gold (fig. 22). Amphibole, primarily actinolite, and epidote are restricted to the lower 100 feet of gravel, chlorite to the lowest 30 feet. Authigenic siderite, present in the lower part of hole A, would be expected in a reducing environment (Pettijohn, 1957, p. 460) such as produced the pyrite.

On the basis of the heavy-mineral suites and detrital gold, the gravel can be separated into four zones, each succeeding lower zone characterized by the addition of one or more new minerals: (1) an upper zone characterized by the presence of ilmenite and zircon, (2) an upper middle zone distinguished by the appearance of pyrite, possibly siderite, less zircon, and increasing gold, (3) a lower middle zone where amphibole and epidote are present with less pyrite and gold values are highest, and (4) a lower zone characterized by the appearance of chlorite. Although holes A and C are

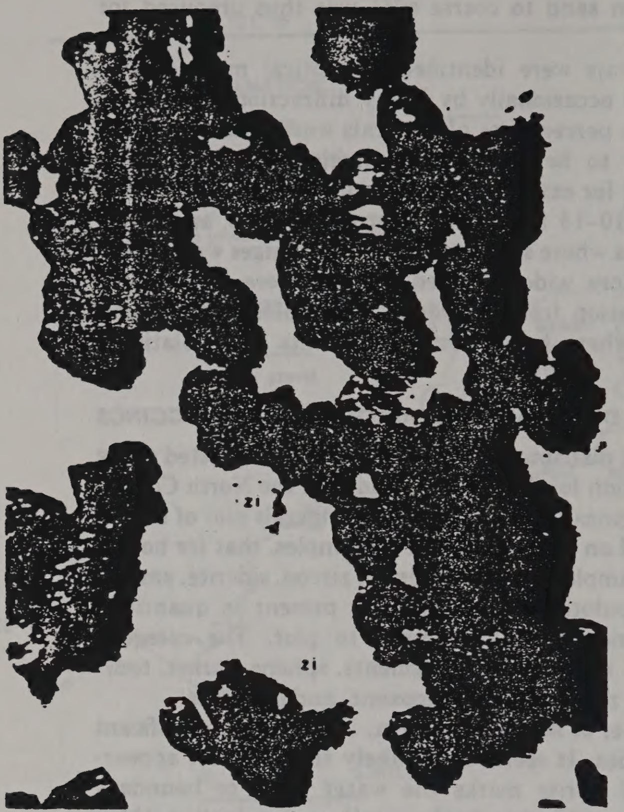


FIGURE 21.—Heavy minerals from upper gravel of hole A in the North Columbia Diggings. Sample collected from a depth of 15 feet. zi, zircon; il, ilmenite.

FIGURE 22.—Heavy minerals from lower gravel of hole A in the North Columbia Diggings. Samples collected from a depth of 365 feet. zi, zircon; il, ilmenite; py, pyrite; si, siderite; ep, epidote; ac, actinolite; G, gold.

separated by no more than half a mile, there is no reason to believe that roughly the same zonation would not be continuous throughout most of the drainage basin.

Detrital platinum was not found in any samples, including surface samples. Chemical analysis of the heavy-mineral concentrates from the lower part of hole A revealed that platinum, palladium, and rhodium were present in negligible amounts varying from 0.1 to 0.02 ppm.

Whitney (1880) reported that "quite a number of diamonds" has been recovered from the gravel in the vicinity of French Corral, the largest 7¼ carats. The occurrence of diamonds within Tertiary gravel from Placer County have also been reported (Mining and Scientific Press, 1870) although the precise location of their occurrence is not known. With this knowledge in mind, the heavy-mineral concentrates obtained for this study were carefully searched for diamonds, but with negative results.

DEPOSITIONAL HISTORY

Even though postdepositional solution and weathering effects have altered some minerals in the gravel, the vertical changes in the heavy minerals can be used to reconstruct the history of deposition. Since the lowest gravels contain the same heavy minerals as the bedrock and in somewhat similar percentages, local source areas probably contributed the bulk of detritus in the early history of the river. Deposition probably was rather rapid, with a minimum of winnowing and reworking, and the rather large percentages of amphibole, epidote, and chlorite indicate only slight chemical weathering. The coarse and poorly sorted texture of the lowest gravel supports this conclusion. The boundary between zones two and three is rather abrupt and seems to represent a major change in depositional characteristics. Only the most resistant and durable heavy minerals, primarily zircon and ilmenite, were deposited in these younger sediments. Euhedral and subhedral zircon and quartz grains are fairly common, implying little transportation and reworking. Rather, chemical weathering is thought to have become a dominant factor in the

breakdown of the rocks. As slope gradients were lowered, soil formation was allowed to continue for longer periods of time without removal by sheetfloods or mass-wasting processes. Only the most physically and chemically resistant minerals would have survived the intense weathering of the subtropical climate believed to have prevailed at that time (MacGinitie, 1941, p. 73).

The presence of pyrite and siderite in zone two does not tell us much about the depositional history of the gravel, because these minerals are postdepositional in origin; however, their presence does imply that organic matter was somehow trapped and buried in the gravel before destruction by oxidation.

Zone two is gradational with zone one. In zone one gold values are the lowest. The amount of zircon increases, and compared to zone two, overall grain size decreases, as clay, silt, and sand are predominant. Source rocks of the sediments deposited here have undergone extensive physical and chemical breakdown.

ROTARY DRILL HOLE SAMPLES, SAN JUAN RIDGE

Heavy minerals are plotted for the gravel portion of holes No. 2, No. 3, and No. 4 on San Juan Ridge (pl. 2). These holes were drilled primarily in an attempt to intersect the deepest, gold-rich part of the gravel; it can be readily seen from the textures and heavy minerals found in all four holes that the objective was not achieved. The textures, for the most part, are typical of the upper bench gravel, similar in some respects to the upper gravel section exposed at Woolsey Flat and Malakoff Diggings (pl. 2). The heavy minerals, with some exception, are also typical of the upper gravel. Ilmenite is ubiquitous; zircon is present in hole No. 3 only. Pyrite and siderite are secondary in origin, as in the drill holes in the North Columbia diggings, and are accounted for by a reducing environment. The entire gravel section is below the water table, because 500–650 feet of volcanic breccia overlies the gravel on San Juan Ridge. The amphibole in all three holes is mostly hornblende, which is similar to that in the overlying andesite breccia (upper part of holes No. 2 and No. 4). The hornblende in these samples is believed to have been derived from higher in the hole by caving, sloughing, and circulation of the mud; therefore, it should not be considered an inherent mineral of the gravel. Other than confirming the "upper bench" character of the gravel in these drill holes, the heavy minerals are not particularly useful. Only a few flakes of gold were found in the samples, and values are less than 1 cent per cubic yard.

SURFACE SAMPLES

Surface exposures of Tertiary gravel were sampled throughout the area in an effort to see if there was a systematic change in mineral suites along the river channel. Generally each sample consisted of two or three 15-inch pans of gravel. Whenever possible samples of both upper and lower gravel were collected at each locality. Heavy-mineral suites from representative samples are shown in figure 23. Generally the heavy minerals are similar to those found in the drill samples in the North Columbia diggings with some exceptions. Magnetite in minor amounts was removed with a hand magnet. As with the drill samples from the North Columbia diggings, the upper gravel is high in ilmenite and zircon, whereas the lower gravel has significant amounts of the less stable minerals, epidote, hornblende, actinolite, pyrite, and sphene. Pyrite was missing in many of the lower gravel samples because the exposures had been oxidized following the mining activities of the past century. Wherever fresh exposures of the lower gravel could be found, pyrite was present. X-ray patterns of samples collected near outcrops of ultramafic intrusive rocks (Liberty Hill and Little York) revealed both chromite and magnesian-chromite ($FeCr_2O_4$ with $MgAl_2O_4$) in the heavy-mineral suite.

No obvious systematic change in the heavy minerals is apparent along the channel in the area studied. The intrusive granitic and ultramafic rocks and the low-grade metamorphic phyllite, slate, quartzite, and schist cropping out in the foothills region most probably were the source for the heavy minerals found.

Heavy minerals from three superimposed textural units within the gravel along Interstate Highway 80 near Gold Run were examined; they are plotted in figure 24. All three units fall within the upper gravel stratigraphically, although the finer grained middle unit contains heavy minerals more characteristic of the lower gravel. It seems that at this particular locality, the normal depositional characteristics of the upper gravel were interrupted and a temporary return to conditions of more rapid deposition of unweathered detritus occurred. During times of heavy precipitation or landsliding, relatively fresh unweathered materials could have been carried into the rivers and locally overwhelmed the more normal sediments being deposited. It seems quite probable that this took place repeatedly throughout the history of the gravel deposition at various places along the length of the river channel.

TRACE OF RESTORED CHANNEL

The trace of the gravel-filled channel as it may have existed near the close of the Eocene is shown in figure 2. The flood-plain width indicated probably represents a minimum width. The flood plain was probably wider than shown in the western half of the area downstream from Badger Hill; this part of the flood plan has under-

gone more erosion than the part farther east, such as at North Columbia, where the retreating cover of volcanic rocks has more recently exposed the channel gravels. The wide expanse of gravel marking the position of alluvial deposition should be thought of as a valley fill developed in an area that had a fairly well integrated drainage system. The rivers were not as wide as the gravel fill, except perhaps in times of extraordinary floods: their channels meandered back and forth across the fill material.

Only that portion of the drainage is shown for which there is reasonably clear evidence for its reconstruction. Several isolated, outlying exposures of gravel cannot be tied definitely into the drainage system.

EVIDENCE FOR RECONSTRUCTING DRAINAGE

The evidence for reconstructing the drainage is primarily the location of preserved gravel deposits. Extensive drilling by private companies in the 1930's between North Columbia and Badger Hill Diggings provided the evidence for the positioning of the channel center in this locality: the center indicated in figure 2 is meant to correspond to the lowest or deepest position within the bedrock channel. In some places, as between North San Juan and French Corral, it is actually an incised trough sometimes referred to as a "gut." Elsewhere, it is merely the lowest point in the overall broad channel depression.

Rotary drilling on San Juan Ridge between Malakoff Diggings and Moore's Flat, together with information from the underground drift mine, the Derbec, provided data for the channel restoration in this area.

Pebble imbrication and crossbedding made possible a reconstruction of the drainage pattern in the southern part of the area mapped. It can be quite clearly shown that gravel in the Gold Run–Dutch Flat area was deposited by water flowing in a northerly direction and that this channel was definitely part of the same drainage system existing farther north in the vicinity of Nevada City and North Columbia. This conclusion was advanced earlier by Lindgren (1911) and Hudson (1955). Imbrication of cobbles and boulders in the lower gravel at Indiana Hill (fig. 25) and Little York (fig. 26) clearly indicate this pattern of current flow.

The present bedrock elevations on the floor of the channel, where exposed, provide further clues for restoration of the drainage system. These data must be tempered with the realization that tilting of the Sierra Nevada and local faulting of the channel gravel have appreciably altered the original slope of the river floor.

MAJOR TRIBUTARY EXTENSIONS BEYOND MAP AREA

Evidence for extension of the drainage system beyond the map boundaries to the northeast is scanty. The high parts of the Sierra Nevada have undergone deep erosion and the volcanic cover on the gravel has been removed from all but a few narrow ridge tops. The main channel, traced only as far as American Hill in this study, is shown by Lindgren (1911) to extend beneath the thin volcanic cap to the northeast, then bend around to the southeast much as the Middle Yuba River does today. The tributary coming in from the vicinity of the Sixteen-to-one mine at Alleghany is covered by volcanic rocks farther north but probably extends as far north as the town of Forest.

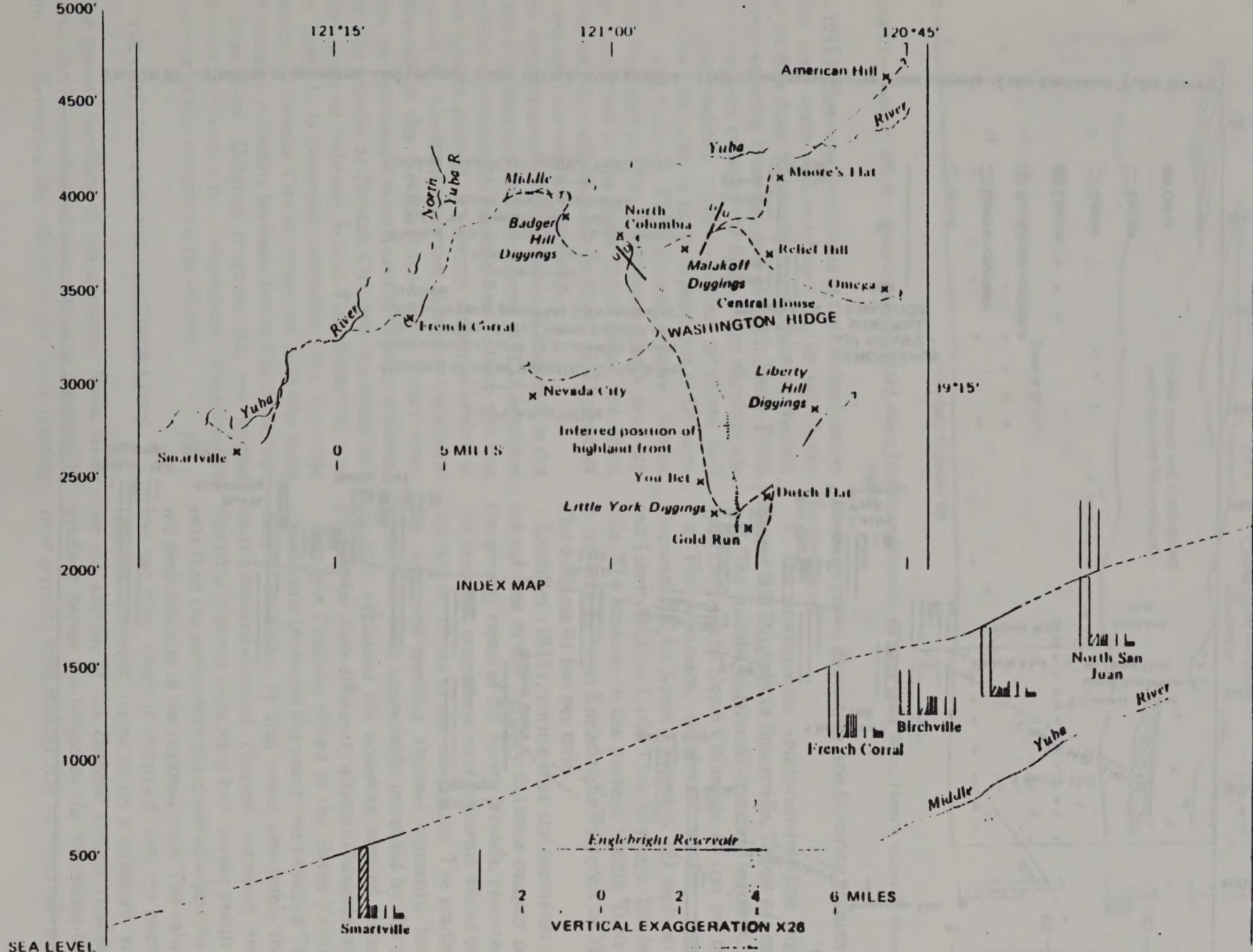
The mapping of this study was not extended south of the North Fork of the American River, although the gravel of this region would allow a much longer drainage restoration (Lindgren, 1911). Below Smartville, west of the map area, the gravel was traced to the present flood plain of the Yuba River. Projecting the present gradient of the ancestral Yuba River farther west places the channel beneath the land surface (fig. 23). Where last exposed near the present Yuba River flood plain, the gravel contains cobbles and boulders that are as much as 8 inches in diameter. From this fact, the river mouth is inferred to have been some distance farther west.

The location of the headwaters and size of the total drainage basin of the ancestral Yuba River will be discussed in a subsequent chapter.

DIFFERING INTERPRETATIONS

Lindgren (1911) showed a major tributary entering the main channel near North San Juan but I do not because the mapping did not reveal evidence for such a tributary. If such a tributary did enter here, the critical area of confluence has been destroyed. Mapping did not extend north as far as Camptonville, the nearest exposure of gravel in this direction.

In the area of the Malakoff hydraulic pit, Lindgren (1900) shows two drainages coming together—one from the north in the Bloody Run Creek area and one from the east, from the Relief Hill tributary. Lindgren's reconstruction shows the channel coming in from Bloody Run Creek bending around to the east and connecting with the exposed gravel at Woolsey, Moore's, and Orleans Flats. Field mapping in 1967 and 1968 revealed that only thin, isolated gravel deposits were present in the Bloody Run Creek area. It was concluded, therefore, that this gravel was deposited by a small tributary rather than by the main river. Furthermore, drilling in 1968 and 1969 established the presence of a thick section of sand and clay beneath the volcanic rocks on San Juan Ridge. As these sediments were believed to represent the marginal or flood-plain deposits of the main river channel, the channel is shown extending from the Malakoff pit beneath the volcanic cover to Woolsey Flat (fig. 2). The Derbec mine (Lind-


5000'
4500'
4000'
3500'
3000'
2500'
2000'
1500'
1000'
500'
SEA LEVEL
121°15'
121°00'
120°45'
39°15'
American Hill
Yuba
River
Moore's Flat
Middle
North
Yuba R
Badger Hill Diggings
North Columbia
Relief Hill
Malakoff Diggings
Omega
Central House
WASHINGTON RIDGE
French Corral
River
Yuba
Smartville
Nevada City
Liberty Hill Diggings
Inferred position of highland front
0
5 MILES
You Bet
Dutch Flat
Little York Diggings
Gold Run
INDEX MAP
North San Juan
Birchville
French Corral
Middle
Yuba
River
Englebright Reservoir
Smartville
2
0
2
4
6 MILES
VERTICAL EXAGGERATION X28

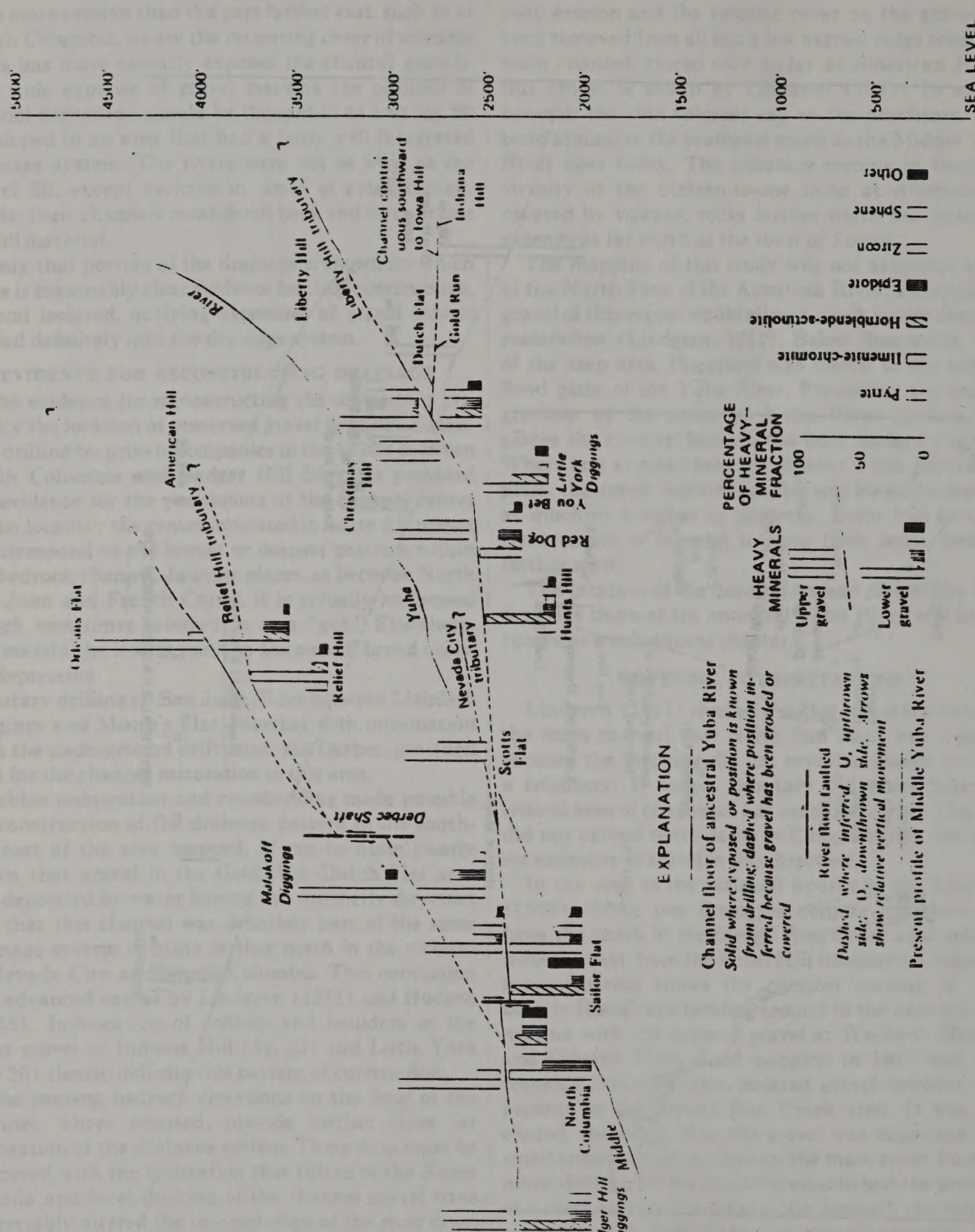


FIGURE 23.—Profiles of ancestral and present Yuba Rivers, with graphs of heavy-mineral suites from gravels of the ancestral Yuba River.

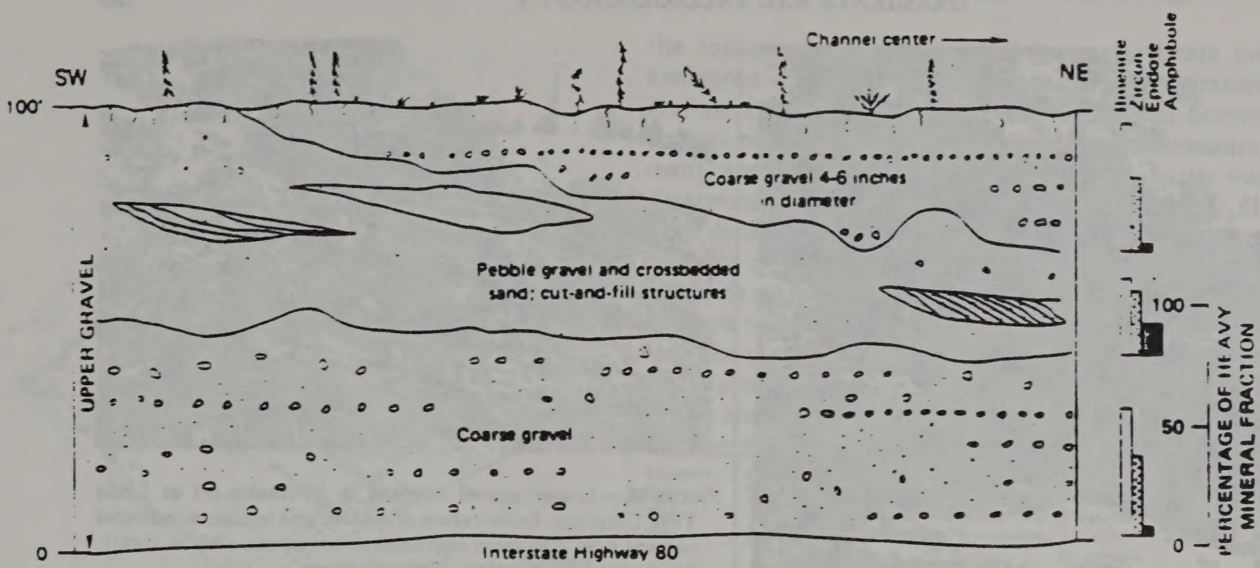


FIGURE 21.—Tertiary gravel (upper gravel) exposed along Interstate Highway 80 near Gold Run. Heavy minerals shown at right.

gren, 1911) allows the positioning of the channel center in that area south and southwest of the drill holes (1–4) on San Juan Ridge (fig. 2). A separate channel is shown entering from the east through Snow Point because the rock types in the Snow Point diggings are all of local bedrock types—siliceous phyllites and slates. The rock types in the hydraulic pits at Orleans, Moore's, and Woolsey Flats include granite, diorite, amphibolite, and phyllite, implying a source area different from that of the Snow Point area. The exact position of the confluence of the Relief Hill tributary entering from the south is uncertain.

The reconstruction of the channel or channels in the Nevada City area is difficult. Lindgren (1911) describes a channel configuration much different from the one shown here in figure 2. A westerly direction of flow is indicated by Lindgren with two separate tributaries originating in the Nevada City–Grass Valley area. Except for two narrow occurrences of volcanic colluvial cover, the gravels at Nevada City are traceable into gravels exposed between Scotts Flat Reservoir and Blue Tent. It is therefore concluded that the wide tributary at Nevada City is connected with the main river channel extending between Scotts Flat Reservoir and Blue Tent. Detailed mapping did not extend southwest of Nevada City; consequently, the complete restoration of this tributary is not shown.

GRADIENTS AND PALEOGEOGRAPHY

The present gradient of the ancestral Yuba River channel can be determined from the data compiled in figure 23. Removing the effects of the local faulting after the gravel deposition, the following gradients can be measured:

1. The downstream, southwest-trending segment, Badger Hill Diggings to Smartville, 79 feet per mile.
2. The middle, northwest-trending segment, Little York Diggings to North Columbia, 17 feet per mile.
3. The upstream, southwest-trending segments, American Hill to North Columbia, 121 feet per mile; and Liberty Hill to Little York Diggings, 120 feet per mile. The gradient of the present Middle Yuba River in the area between Englebright Reservoir to Moore's Flat averages 64 feet per mile.

Lindgren (1911), noting that the northwest-trending portions of the Tertiary channels roughly parallel the present trend of the Sierra Nevada, reasoned that the gradients from these river segments would be little changed by tilting of the mountains. The gradients of the southwest-trending channel segments, however, would have been appreciably increased by the tilting. He thus explained the observed marked changes in gradients of these differently directed channel segments.

The late Cenozoic tilting of the Sierra Nevada has significantly increased the southwest-trending Tertiary channel gradients. It also seems reasonable that the southwestward-directed channel segments trending down the regional slope have had steeper pretilt gradients than the northwestward-directed segments trending perpendicular to the regional slope. The value of 17 feet per mile, which is obtained from the northwest-trending segments, seems to be a minimum gradient for the portion of the channel system studied. The gradient before the tilting of the southwest-trending river segment from Badger Hill to Smartville probably

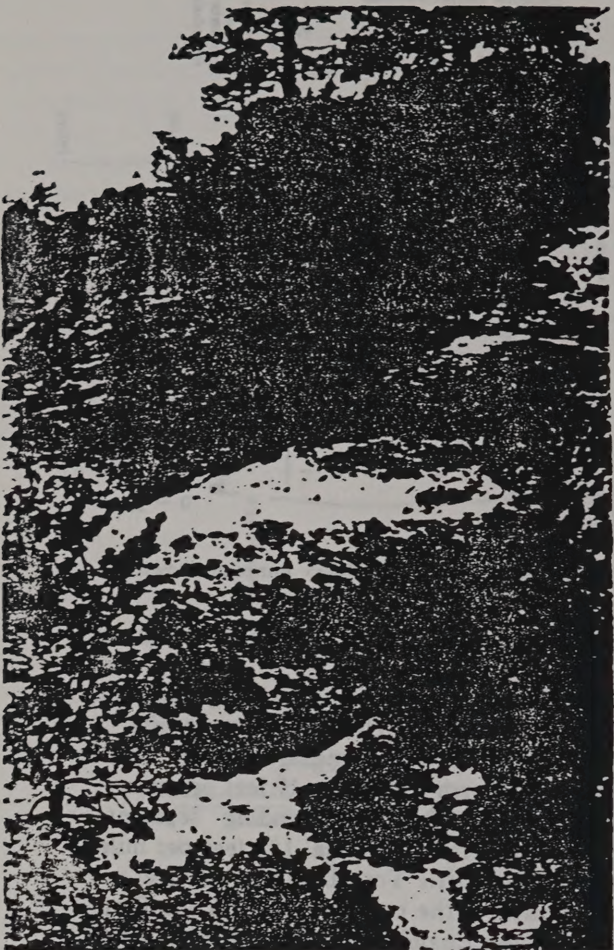


FIGURE 25.—Lower gravel exposed on east side of hydraulic pit at Indiana Hill. Note imbrication in coarse lower gravel indicating current flow right to left (south to north). See figure 2 for location of photograph. View east.

was slightly greater than 17 feet per mile because it was flowing down the regional slope; here a gradient of 20–25 feet per mile would seem reasonable. By comparing this gradient with the present gradient of 79 feet per mile, the effect of tilt after the gravel deposition measured along this stretch of the channel is 54–59 feet per mile. The amount of straight-line tilt measured perpendicular to the mountain axis would be 66–72 feet per mile. If it can be assumed that a similar amount of tilt affected the entire drainage system, then the upstream, southwest-trending portions of the ancestral Yuba River must have had pretilt gradients of 60–65 feet per mile. This gradient is similar to gradients of the present Middle Yuba River in this area.

The relatively steep gradients before the tilt northeastward from the North Columbia and Little York Diggings areas suggest that the river between the two

FIGURE 26.—Lower gravel exposed in hydraulic pit at Little York Diggings. Imbrication of cobbles and boulders indicates current flow was from southeast to northwest (left to right). See figure 2 for location. View southwest.

areas flowed parallel to the base of an eastward-retreating highland front. Evidence supporting the presence of such a highland front is manifest in the abrupt change in bedrock slope below the volcanic rocks on Washington Ridge between Little York Diggings and North Columbia (fig. 23). At Central House along Highway 20, the volcanic breccia is about 200 feet thick. One mile west the volcanic breccia has increased in thickness to about 500 feet owing to a lowering of the bedrock surface to the west (fig. 27). A block diagram showing the inferred paleogeography of part of the ancestral Yuba River drainage is shown in figure 28. The river between Little York Diggings and North Columbia probably occupied a broad, fairly flat trough at the foot of a highland and flowed northwest, generally parallel to the dominant structural trend of the metamorphic bedrock.

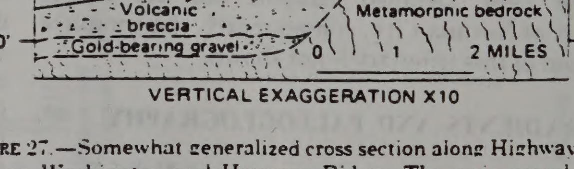


FIGURE 27.—Somewhat generalized cross section along Highway 20 on Washington and Harmony Ridges. The main gravel-filled channel trends northwest, nearly at right angles to this section, and crosses at the base of the bedrock highland front to the east. Line of section shown on plate 1.

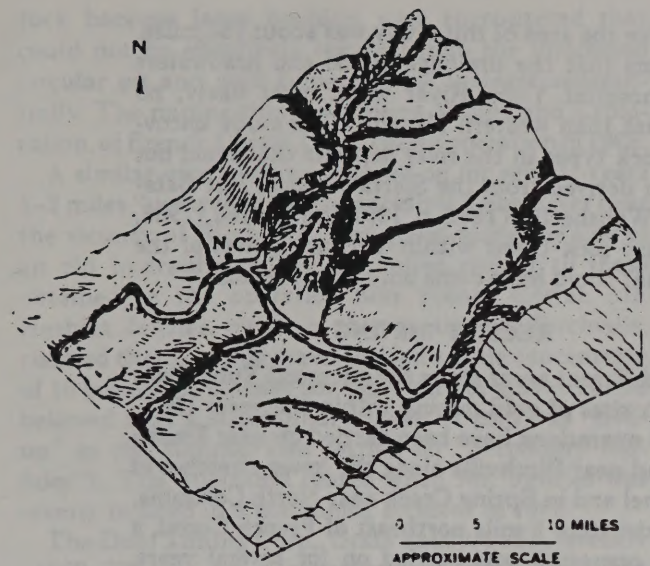


FIGURE 28.—Generalized block diagram of inferred paleogeography of part of the ancestral Yuba River drainage. Vertical scale is partially exaggerated. N.C. = North Carolina: L.Y. = Little York Diggings.

The anomalous reverse gradients near Badger Hill Diggings and Gold Run are probably a result of tilting of the Sierra Nevada. Because the channel trends northeast in these two locations, the more downstream segments of the channel experienced greater uplift.

DRAINAGE AREA AND LENGTH OF THE ANCESTRAL YUBA RIVER

Although the gradient and width of the ancestral Yuba channel can be determined with a reasonable degree of certainty, the original length or drainage area of the old river system cannot be directly measured. Because little remains of the river deposits above the 4,000-foot level, attempting to tie together widely scattered gravel outcrops into an accurate drainage reconstruction is practically impossible. By comparing modern river systems, it was hoped that an approximation of the unknown parameters of length and drainage area could be determined. A comparison of gradient, valley width, and drainage area might show a relation such that, given a particular gradient, a drainage size could be predicted within somewhat limited values.

A modern analogue of the ancestral Yuba River would be located in a warm, wet climate with moderate relief. Southern Mexico fits these conditions, but the lack of detailed topographic maps precludes making adequate comparisons. The southeastern part of the United States approximates the right climate, but the characteristic relief conditions are lacking. The river systems selected for study are those of western Washington, Oregon, and northern California, because both the topographic conditions and the wet climate are analogous. The elements lacking are warm temperature and semitropical vegetation. U.S. Geological Survey stream-gaging stations were used as points of measurement above which the size of drainage basin was determined. River gradients and width of valley fill were determined at the gaging stations, and only those portions of rivers that occupied a valley 500 or more feet wide were selected for study. Restricting the study to these valleys best simulated an aggrading river system like the one that deposited the gravel in the ancestral Yuba River valley. The following rivers were used in this study:

Northern California		*Oregon*	*Washington*		
Klamath	Sacramento	Deschutes	Hoh	Toutle	Snoqualmie
Eel	Trinity	Umpqua	Humptulips	Cowlitz	Skagit
Mad	American	Santiam	Satsop	Nisqually	Nooksack
Pit	Feather	Willamette	Wynoochee	Puyallup	Naches
			Lewis	Duwamish	

The study included about 65 different measurement localities and showed a great variability both within and between the different river systems. Particularly evident was the great variation in the width of valley fill for the rivers with similar slope and drainage size.

Logarithmic plots of channel gradient versus drainage area and channel gradient versus valley-fill width, shown in figures 29 and 30, show the broad limit possible for any given channel gradient rather than specific values of drainage area and channel width. For example, using the determined value for pretilt gradient of the ancestral Yuba River of 20–25 feet per mile, the plots show that the largest area likely for the drainage basin would be 1,800–2,500 square miles. Lindgren (1911)

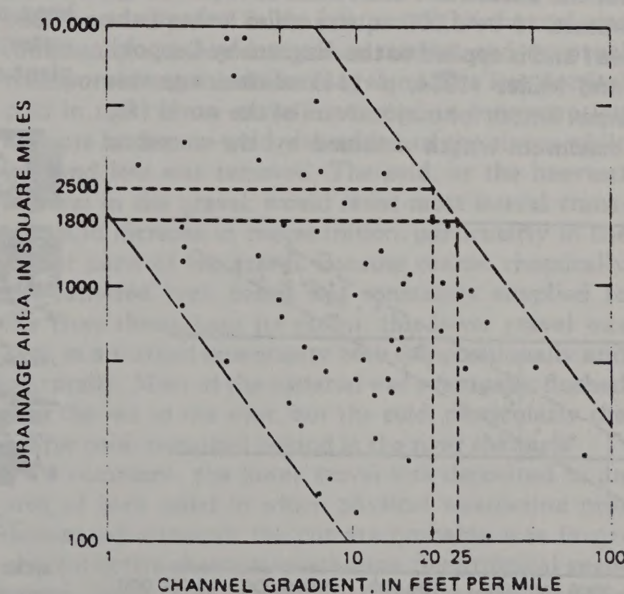


FIGURE 29.—Relation between channel gradient and drainage area for selected localities on rivers in the Pacific States.

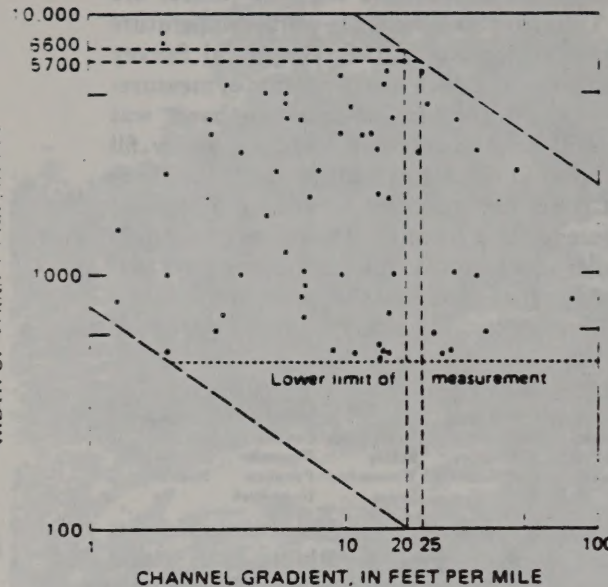


FIGURE 30.—Relation between channel gradient and width of valley fill for selected localities along aggrading rivers in the Pacific States.

estimated a minimum drainage basin for the ancestral Yuba of approximately 1,000 square miles. Using the same 20–25 feet per mile gradient and applying it to the plot of valley-fill width versus channel gradient (fig. 30), the valley-fill widths would likely range from about 500 feet to as much as 6,600 feet. This width corresponds well with the variation of valley-fill widths observed for the ancestral Yuba River. If the drainage area is assumed to be 2,000 square miles (above the area studied) and is applied to the diagram by Leopold, Wolman, and Miller (1964, p. 144) of drainage area versus channel length for major rivers of the world (fig. 31), the maximum length attained by the ancestral Yuba above the area of this study was about 150 miles. This means that the drainage divide and headwaters of the ancestral Yuba River were, most likely, no farther east than western Nevada. This study uncovered no rock types in the river deposits that could not have been derived from the Sierra Nevada. But Bateman and Wahrhaftig (1966, p. 139) and Durrell (1966, p. 187) list rock types within Tertiary gravel of the Sierras that could have come only from Nevada.

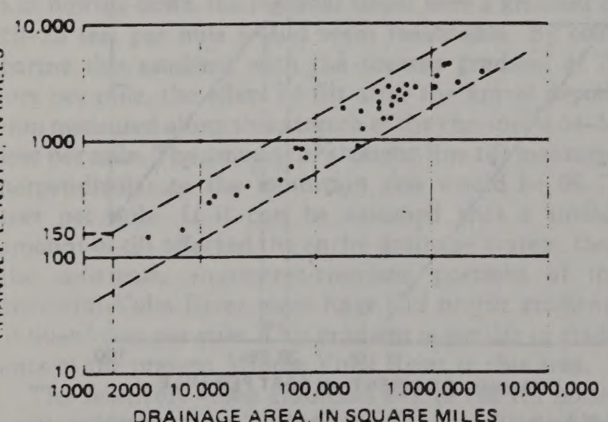


FIGURE 31.—Relation of channel length to drainage area for various rivers of the world (after Leopold and others, 1964).

RECENT MINING

Three different areas along the ancestral Yuba River have been sites of gold mining within the years 1967–70. Small operations have been carried on near French Corral and near Birchville along the lower stretches of the channel and in Spring Creek near North Columbia.

Approximately 1 mile northeast of French Corral, a two-man operation was carried on for several years, processing lower gravel within an old hydraulic pit (fig. 32). A large bulldozer was used to break up the gravel and to move it within reach of a bucket dragline. The gravel was washed through a ¾-inch trommel, and gold was collected in both hungarian riffles and a rotating bowl. Electric power supplied by a portable generator was used to run the concentrator. The plant capacity of 900 cubic yards of gravel per day was seldom achieved because of constant equipment breakdown. "Clean up" once a week produced values ranging from $7 to $0.70 per cubic yard that was taken from blue gravel 20–70 feet above bedrock. The marginal operation barely paid expenses, it was continued with the hope of reaching richer gravel with depth. A limited water supply restricted the length of time that the plant could operate. Costs became excessive near bed-

FIGURE 32.—A small gold-mining operation located in hydraulic pit of the ancestral Yuba River 1 mile northeast of French Corral. Blue gravel is exposed in a hole in the foreground where the water table was intersected. Note darker colored, wet ground.

rock because large boulders were encountered that could not be effectively removed from the undrained circular pit and gold values did not increase substantially. The mining company, Sierra Mining and Exploration, of French Corral, Calif., ceased operation in 1968.

A similar operation was carried on for several years 1–2 miles "upstream" on the ancestral Yuba River near the vicinity of Birchville. Blue gravel along the side of an old hydraulic pit yielded values similar to those obtained in the operation near French Corral. Mr. Herbert Jeffries, owner of the operation at Birchville, claimed that the blue gravel yielded a table concentrate of 10 pounds of sulfides per cubic yard of gravel. It was believed that a substantial amount of gold was "tied-up" in the sulfides (see section on "Secondary Sulfides"). The operation ceased when Mr. Jeffries was severly injured in a nonmining accident in 1968.

The Dulo Mining Corp., Grass Valley, Calif., recently (1969–70) set up a concentrating plant along the ancestral Yuba River on Spring Creek (fig. 33), approximately 1–2 miles southeast of North Columbia at a position just upstream from the confluence of the two major branches of the ancestral Yuba River where the tributary is narrow. The operation is similar to those previously described, except that the equipment is somewhat larger and can handle a greater volume of gravel. An "air-flow" table is used to separate the gold from the heavy black sand concentrate. Lower gravel approximately 80 feet above bedrock is being mined. The operation is in its incipient stages, and gold values have not yet been determined.

GEOLOGIC HISTORY

The foothills region of the Sierra Nevada was part of a north-trending highland that experienced deep and pervasive erosion during the Cretaceous and earliest Tertiary (Bateman and Wahrhaftig, 1966, p. 126–127). The great amounts of sediment eroded from this highland accumulated in the Pacific geosyncline to the west. A layer of rock possibly as much as 17 miles thick was removed from the area we now call the Sierra Nevada, and the great batholith forming the core of the mountains was unroofed and exposed (Bateman and Wahrhaftig, 1966, p. 127–128). For the highland to persist throughout the Cretaceous and early Tertiary, the area probably was more or less continually uplifted. Perhaps the earliest coarse gravels of the ancestral Yuba River reflect a particularly vigorous pulse of uplift. Most of the erosion had already taken place by the time the ancestral Yuba River was carving its channel. The final position of that drainage system can be reconstructed, but the older drainage patterns may never be known. The final drainage system was established prior to the middle Eocene and perhaps as early as the Paleocene. In its early history, the system probably was characterized by narrow, high-gradient high-energy streams and rivers flowing in deep, narrow steep-walled canyons much as the modern rivers that drain the Sierras. These rivers possessed sufficient energy to transport 10-foot boulders a minimum distance of 8 miles.

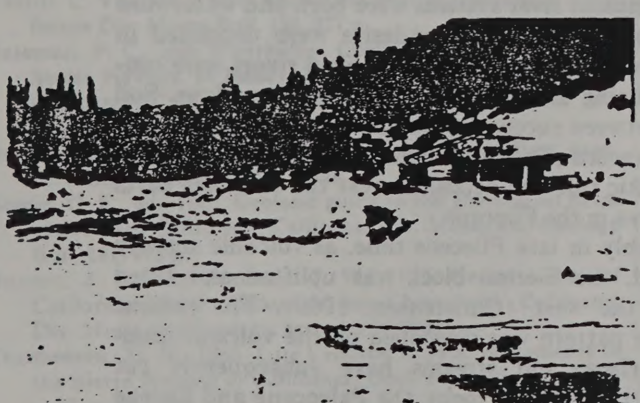


FIGURE 33.—Mining the lower gravel of the ancestral Yuba River along Spring Creek.

During the long period of degradation, local gold-bearing rocks were being eroded, physically broken down, and transported by the rivers. The gold was released from the host rocks mainly by mechanical breakdown but partly by chemical breakdown. As the rivers were primarily downcutting throughout their early history, no great thickness of fill accumulated in the valleys. Perhaps as much as 50–100 feet of coarse, poorly sorted gravel in the bottom of the channel was continually subjected to scour, mixing, and eventual transport along the drainage system. The free detrital gold in this "lower gravel" increased in concentration as more and more gold was added to the rivers while less and less was removed. The gold, as the heaviest mineral in the gravel, would resist most lateral transport and increase in concentration, particularly in the lowest parts of the gravel. Because coarse, chemically unweathered rock debris was constantly supplied to the river throughout its extent, this lower gravel was kept in a state of immaturity both compositionally and texturally. Most of the material was eventually flushed into the sea to the west, but the gold, particularly the coarse gold, remained behind in the river channels.

In summary, the lower gravel was deposited in an area of high relief in which physical weathering predominated although the climate probably was favorable for active chemical weathering. Semitropical vegetation, large trees in particular, was carried into the rivers and buried with the gravel. The rivers continued to downcut, predominantly during times of flooding

and vigorous runoff, and the 50- to 100-foot-thick veneer of coarse gravel flooring the river valley would be moved downvalley only to be replaced by new material from upstream.

As downcutting proceeded, the river systems became integrated and through headward erosion extended their basins and tributaries eastward. By middle or late Eocene time, the steep slopes of the earlier landscape had given way to gentle slopes. Relief was subdued and chemical weathering was the dominant process in the breakdown of the rocks. Thick soils developed on the land surface, and all but the most resistant rocks and minerals were broken down to their constituent elements and either carried away in solution or recombined into stable clay minerals. The river gradients became less steep as dissection lowered the drainage basin until the final phase of river aggradation began. Subangular to subrounded milky-white quartz pebbles and euhedral and subhedral zircon grains deposited in the rivers during this period imply that the material eroded was not transported for a great distance. Local sources continued to supply the detritus as before, but because of the intensity of the weathering, only the most resistant minerals survived the journey to the rivers. The valley fill increased in thickness as the rivers continued to aggrade, and extensive flood plains were formed. The rivers, perhaps 150 miles in length, may have been draining areas as far away as western Nevada. Plant leaves were occasionally trapped, and their impressions were preserved in the clays deposited on the flood plain. MacGinitie (1941, p. 78) characterizes the climate as probably similar to the present climate in the lower slopes of the Sierra Madre in the State of Vera Cruz, Mexico—a frostless subtropical climate with heavy rainfall in the warm season and a well-marked dry season. The average annual temperature at the low altitudes may have been 65°F, the annual rainfall at low altitudes, 60 inches.

Gold-bearing rocks continued to be eroded and broken down, but little of the gold was preserved in the gravels and sands deposited during this aggradational phase of the river's history. The lack of gold in these rocks poses a puzzling and unsolved problem. Judged by the amount of vein quartz in the gravels, tremendous amounts of potentially gold-rich rocks were eroded. It is not known whether the gold lagged behind in the regolith and was not transported in any major quantity, or was broken down physically to a small size and transported into the bordering oceans, or was carried off in solution. Further exploration of sedimentary rocks of equivalent age in the Central Valley, along with laboratory experiments on physical and chemical breakdown of gold, will aid in solving this problem.

These conditions of low relief, intense chemical weathering, aggrading rivers with wide flood plains and a subtropical climate continued to the end of the Eocene, after which volcanic activity began.

At the beginning of the Oligocene Epoch, volcanoes erupted, probably to the east near the drainage heads. Rhyolitic tuffs were deposited over much of the area extending to the bordering ocean on the west. These tuffs have been preserved most commonly in the river valleys along the flood plains where they were quickly covered by younger volcanic rocks. River gradients were increased, perhaps by renewed uplift, and volcanic cobble gravel was deposited within the older river valleys. A few of the older prevolcanic channels were filled with the volcanic detritus, and a thin blanket of volcanic clastic rocks covered parts of the old land surface. Channels of new drainage patterns formed, occasionally cutting across and eroding the older gold-bearing gravel, reworking the older alluvial deposits and becoming enriched in gold.

The climate probably was very similar to that of the middle Eocene. Fossil flora from volcanic rocks of this age imply a climate intermediate between temperate and tropical (Potbury, 1937). Volcanic eruptions, together with alluvial erosion and deposition, continued throughout the Oligocene to Pliocene.

Widespread volcanic mudflows covered most of the landscape during the Miocene and Pliocene. These predominantly andesitic flows probably were derived from the slopes of volcanoes located near the present crest of the Sierras. Detritus repeatedly flowed down the regional slope toward the west and extended out into the Central Valley. The flows were initially confined to the unfilled major river valleys that had been formed in the Eocene. As these became filled, the mudflows spread out and covered all but the highest hills of the surrounding land surface. Between successive mudflows incipient river systems were born and waterworn boulders and cobbles of andesite were deposited in restricted channels. These short-lived rivers were constantly being destroyed by subsequent mudflows. Soil zones between successive mudflows, though rarely preserved, record periods of diminished volcanic activity. A rhyolitic tuff was deposited over the southern parts of the area in the Pliocene.

Probably in late Pliocene time, as volcanic activity subsided, the Sierran block was uplifted and tilted toward the west (Christensen, 1966). The present drainage pattern was developed on the volcanic mudflow surface, and streams have subsequently cut through the volcanic rocks, the Paleocene and Eocene gold-bearing gravel, and several thousand feet into bedrock. The Sierran block must have been tilted west of the present foothill–Central Valley boundary, for

the ancestral Yuba River has steep tilt-produced gradients several miles west of the foothill front (the last appearance of the gravel before it disappears beneath the present land surface). Associated with the tilting, small-scale faulting produced displacements of the prevolcanic gravel and overlying volcanic rocks. During and following the uplift, initiated in late Pliocene time, most of the volcanic cover was eroded and vast areas of the prevolcanic land surface exhumed. It is because of this exhumation of the early Tertiary surface that major portions of the prevolcanic gold-bearing gravel is exposed today. The modern rivers cut through the old gravel deposits and isolate them on the interstream divides. Long stretches of the prevolcanic gravel were eroded in places where the modern rivers followed the earlier river courses; the reworked gold became part of the present river deposits, and much of it has been subsequently mined.

Although Pleistocene glaciers were not present in the area of study, the associated wet cool climate probably promoted extensive mass wasting and the reduction of the volcanic deposits capping ridges. Colluvial deposits continued to accumulate at the base of the oversteepened volcanic cliffs. However, the greatest recent change of the land surface was produced by man. One cannot help but be overwhelmed by the enormous effect of hydraulic mining on the landscape. Hydraulic mining, by disrupting the natural geologic processes, locally changed the foothills landscape as well as the present stream and river systems. Although this happened decades ago, the effects are still vividly apparent in the foothills and the Central Valley, and in San Francisco Bay.

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INDEX

[Italic page numbers indicate major references]

INDEX

INDEX

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(Tertiary Gravels/Badger Hill)

Report of Investigations 7935

Drilling and Sampling Tertiary Gold-Bearing Gravels at Badger Hill, Nevada County, Calif.

By Russell R. McLellan, Richard D Berkenkotter, Richard C. Wilmot, and Robert L. Stahl
Division of Mine Systems Engineering, Denver, Colo.

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TABLES

DRILLING AND SAMPLING TERTIARY GOLD-BEARING GRAVELS AT BADGER HILL, NEVADA COUNTY, CALIF.

by

Russell R. McLellan,[1] Richard D. Berkenkotter,[1] Richard C. Wilmot,[1] and Robert L. Stahl[2]

ABSTRACT

As part of an effort to develop new or improved placer mining technology in the Tertiary gold-bearing channel gravel of northern California, the Bureau of Mines conducted investigations at the old Badger Hill placer mine in Nevada County, Calif., to collect data and information regarding drilling and sampling methods and deposit characteristics.

Sampling studies involved the collection and processing of 400 tons of large-and small-volume samples from drill holes, underground workings, surface pits, and blasthole rounds. A total of 355 samples averaging 2,296 pounds per sample provided reasonably accurate data for analysis of the gold distribution. These studies indicated that the bulk of the gold is within 40 feet of bedrock in stacked, lenticular zones of cemented gravel that are largely confined to the relatively narrow, meandering course of the deepest portion of the bedrock channel.

Drilling studies involved the testing of truck-mounted rotary, bucket, and vibratory drills, none of which proved to be a completely versatile tool for sampling quickly and reliably both consolidated and unconsolidated gravel.

INTRODUCTION

To meet the national requirements for increased domestic gold production, the Bureau of Mines, under the aegis of the Heavy Metals Program of 1966-70, conducted a nationwide search for favorable gold-bearing deposits. Preliminary investigations indicated that the immense Tertiary channel gravel deposits of California (1, 8)[3] contain one of the largest reserves of gold in the United States; reserves that might be developed rapidly with the application of new or improved placer mining technology. The Badger Hill Deposit in Nevada County, Calif., was selected as a typical and convenient site for Bureau of Mines research groups to conduct surface and underground investigations in the Tertiary channel environment.

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[1] Mining engineer (now with Mining Enforcement and Safety Administration, Denver, Colo.).
[2] Geologist (now with Denver Mining Research Center, Denver, Colo.)
[3] Underlined numbers in parentheses refer to items in the list of references preceding the appendix.

Five Bureau of Mines research centers and laboratories participated in mining subsystem studies at the old Badger Hill hydraulic mine, Nevada County, Calif., to identify the problem areas requiring immediate solution to stimulate gold mining in these extensive gravel deposits. The original objective of the program was to delineate a segment of a typical Tertiary channel and then conduct intensive mining research, followed by demonstration mining. Reduced funding and subsequent reorientation of the work resulted in the emphasis being shifted towards research within the problem areas that were encountered for each mining subsystem. The demonstration mining stage of the project was eliminated.

Deposit delineation activities under the Denver Mine Systems Engineering Group were oriented to accomplish the following objectives: (1) Determine the vertical and horizontal distribution of the gold particles; (2) test and improve drilling and sampling equipment and techniques; and (3) provide drilling support for geophysics studies.

Sufficient sampling data were obtained to verify that a relatively normal placer gold distribution exists in the Badger Hill segment of the San Juan Ridge channel. The drill-sampling investigation indicated that the drills tested by the Bureau of Mines were incapable of providing reliable samples of the deep, cemented gravels; however, the rotary drill was found to be essential for rapid probing to bedrock in support of the seismic technique that was developed to establish bedrock configuration.

Throughout this report, sample results are presented in terms of milligrams of gold per cubic yard (mg/cu yd) of in-place gravel and milligrams per ton (mg/ton) of dry gravel. Density tests place white gravel at 2,700 pounds per cubic yard (lb/cu yd), red gravel at 3,570 lb/cu yd, and blue gravel at 3,900 lb/cu yd. To assist with the task of converting from milligrams to the currently fluctuating value of gold, the following items of information are presented:

1. 31,103 mg is equal to 1 troy ounce.

2. Current price of gold per troy ounce divided by 31,103 is equal to the current value of 1 mg of gold at 1,000 fine.

3. The value in item 2 multiplied by 0.9 is equal to the value of 1 mg of gold at 900 fine.

All Badger Hill sample results are presented in milligrams of gold at 900 fine unless otherwise stated.

ACKNOWLEDGMENTS

The Bureau of Mines wishes to thank San Juan Gold Co. for cooperating in this study and for allowing the drilling and sampling to be done on their property. The U.S. Forest Service, the Bureau of Land Management, and property owners in the area aided the study by their interest and cooperation.

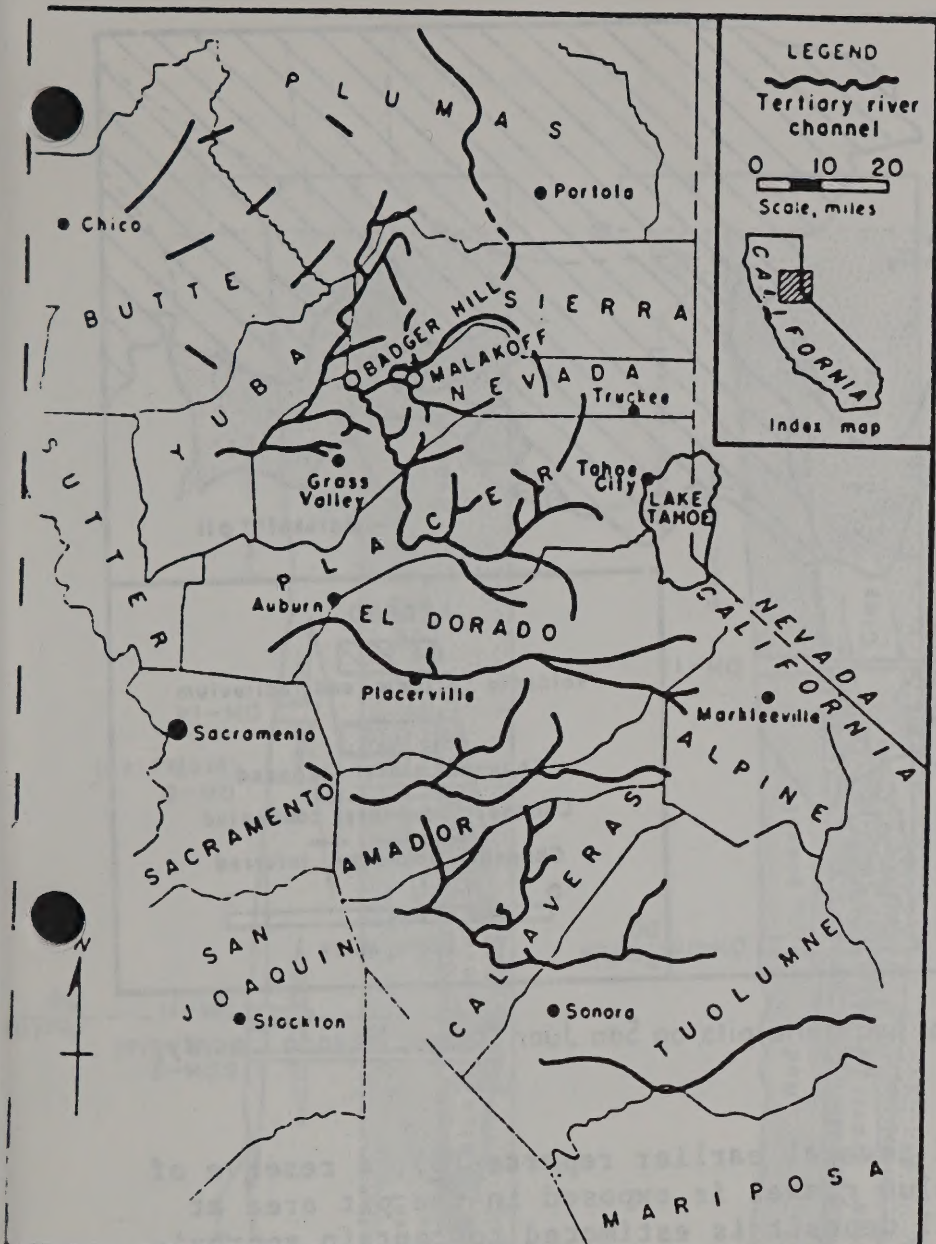

FIGURE 1. - Location of Badger Hill and Malakoff pits, San Juan Ridge, Nevada County, Calif.

DESCRIPTION OF BADGER HILL SITE AND DEPOSIT

The Badger Hill deposit is on the west slope of the Sierra Nevada Mountains, in Nevada County in northern California, approximately 12 miles due north of the town of Grass Valley and 6 miles west of the huge Malakoff pit (fig. 1). The area may be reached by traveling northwest from Grass Valley on State Highway 49 for approximately 20 miles, then northeast on Tyler Road for 5 miles, then north on a dirt road about 1-1/2 miles to the project site, located in sec. 36, T 18 N, R 9 E. The famous hydraulic pits of North Columbia and Malakoff (5-6), among the largest in the Sierra Nevada, are located a few miles to the east (fig. 2). In dry weather, access to the area is excellent; however, during the rainy season the dirt road leading to the site often is excessively muddy.

The deposit is perched on San Juan Ridge from 700 to 900 feet above and between the Middle and South Yuba Rivers. Approximately 5,000 feet of the north end of the deposit was mined hydraulically (3) in two benches prior to 1884, creating a mile-long pit known as the Badger Hill diggings. A vertical bank of gravel approximately 120 feet high separates the lower from the upper workings. The lower pit was hydraulicked to bedrock, laying bare about 2,000 feet of the channel bottom. Total relief in the pit area is about 360 feet; the altitudes range from 2,340 feet in the lower pit to about 2,700 feet on the high point of the upper bench (fig. 3, in pocket).

Exploitation of the Badger Hill deposit and of all other similar deposits in northern California by large-scale hydraulic mining ceased in 1884 because of legal restrictions placed upon the disposal of debris (4). Production figures are not available for the Badger Hill diggings, although the pit size indicates that several million cubic yards of gravel were removed prior to

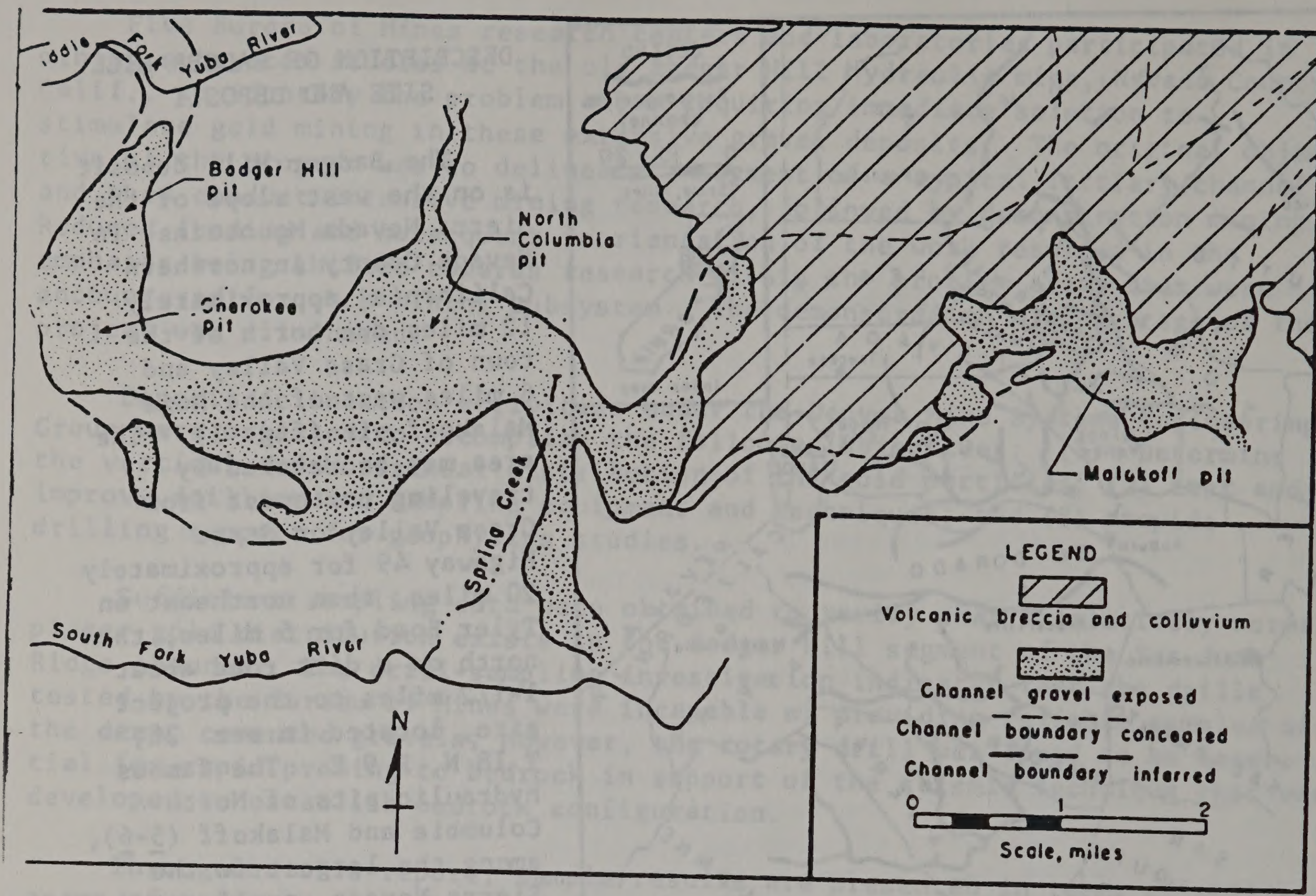

FIGURE 2. - Tertiary channel and major hydraulic pits on San Juan Ridge, Nevada County, Calif.

'884. According to a summation of several earlier reports (5), a reserve of ,700,000 cubic yards of red and blue gravel is exposed in the pit area at Badger Hill. The total Badger Hill deposit is estimated to contain approxi- ately 76,900,000 cubic yards of gravel. Detailed information concerning the adger Hill deposit is lacking, although early engineering reports (5, 7, 9, 11) are available describing the mining, sampling, and reserves of the bulk of he channel on San Juan Ridge.

The Badger Hill test site is at the west extremity f a 6-mile-long egment of Tertiary channel fill that ranges in depth from about 250 feet at adger Hill to about 500 feet at the eastern end of the segment. The channel vidth ranges from 1,200 feet at Badger Hill to approximately 7,000 feet about miles to the east.

The fill is classified into upper and lower gravels. The upper gravel ontains abundant milky-white quartz pebbles interbedded with large amounts of sand and clay. This material is well exposed in the walls of the hydraulic it and normally comprises the bulk of the channel fill in undisturbed areas. ee figures 4 and 5 (fig. 5 in pocket) for geologic cross sections through the adger Hill pit. For explanation of profiles AA', BB', CC', and DD', see igure 3 (in pocket). Within the lower gravel, two units are recognized as

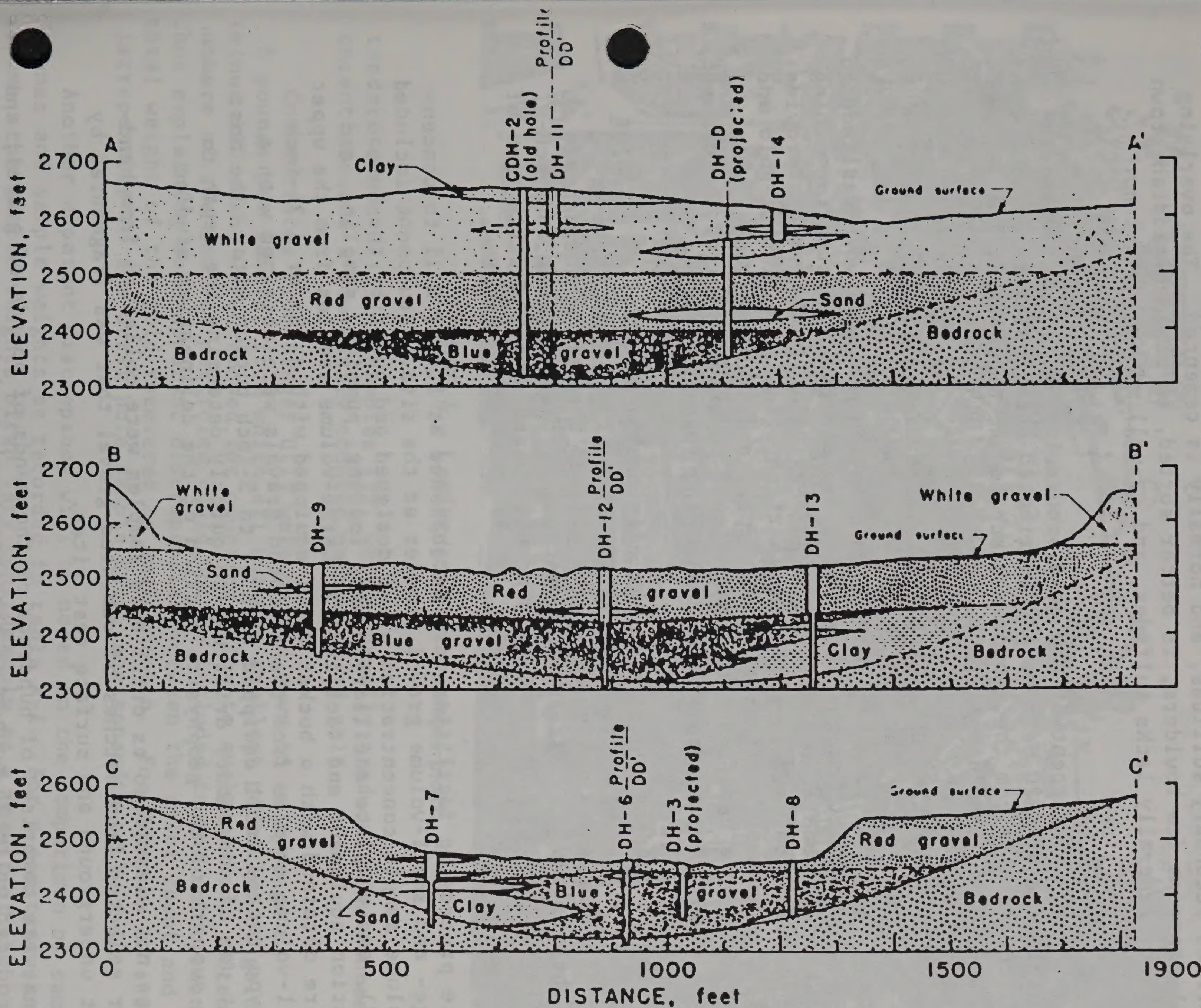

FIGURE 4. - Cross sections through Badger Hill hydraulic pit.

blue gravel and red gravel. The coarse, poorly sorted blue gravel that commonly fills the bottom of the channel is blue-gray in color below the water table, carries secondary sulfides, and normally is cemented. The overlying red gravel contains few boulders, is better sorted, is stained reddish-brown by iron oxide, generally lacks sulfides, and is well compacted but not cemented.

Approximately 80 percent of the gold occurs in the blue gravels near bedrock. The remaining 20 percent is distributed erratically throughout the red and white gravels from the ground surface to the blue gravel contact and from bank to bank in the channel fill.

DRILLING AND SAMPLING DELINEATION COMPLETED BY THE BUREAU OF MINES

The original drill-sampling and trench-sampling projects were designed to determine the suitability of the Badger Hill deposit for demonstration mining. If results were favorable, the drilling and sampling projects were to provide sufficient delineation data (vertical and horizontal distribution of gold and material types, bedrock configuration and depth, and accurate topographic control) to develop a detailed three-dimensional physical model of the deposit, which would be used to develop an optimum mining system.

Reduction of funding resulted in the elimination of detailed drilling that would more firmly establish the distribution of the gold; however, sufficient drilling and sampling were accomplished throughout the deposit to determine the various weaknesses and advantages of the equipment and techniques employed and to verify certain concepts regarding the distribution of gold.

Sample processing facilities were established at Badger Hill to concentrate large- and small-volume gravel samples at the site. Equipment included a large-volume sample concentration plant designed and built by the Bureau (figs. 6-7), and complete ancillary power, loading, pumping, sampling, and transportation equipment and facilities. Large-volume samples from the upper gravels were obtained with a bucket drill equipped with a 30- or a 36-inch bit. Small-volume samples from the cemented gravels were obtained with a standard-type rotary drill equipped with 4- to 5-inch button bits. The most significant use of the rotary drill was to quickly determine the depth to bedrock in support of the geophysics portion of the delineation studies.

Samples not classified as drill hole samples were collected from hand-dug pits or selected from batches of cemented gravel that was fragmented by blasting at underground and surface test sites.

Samples were comprised of hundreds or thousands of pounds of gravel, and normally produced up to 25 pounds of concentrate that might contain from a few dozen to many hundreds of gold particles or colors ranging in size from microscopic specks to flakes 3 mm or more in diameter. The relatively large amounts of concentrate and gold per sample required the adaptation of field evaluation and recording techniques that differed from those employed for

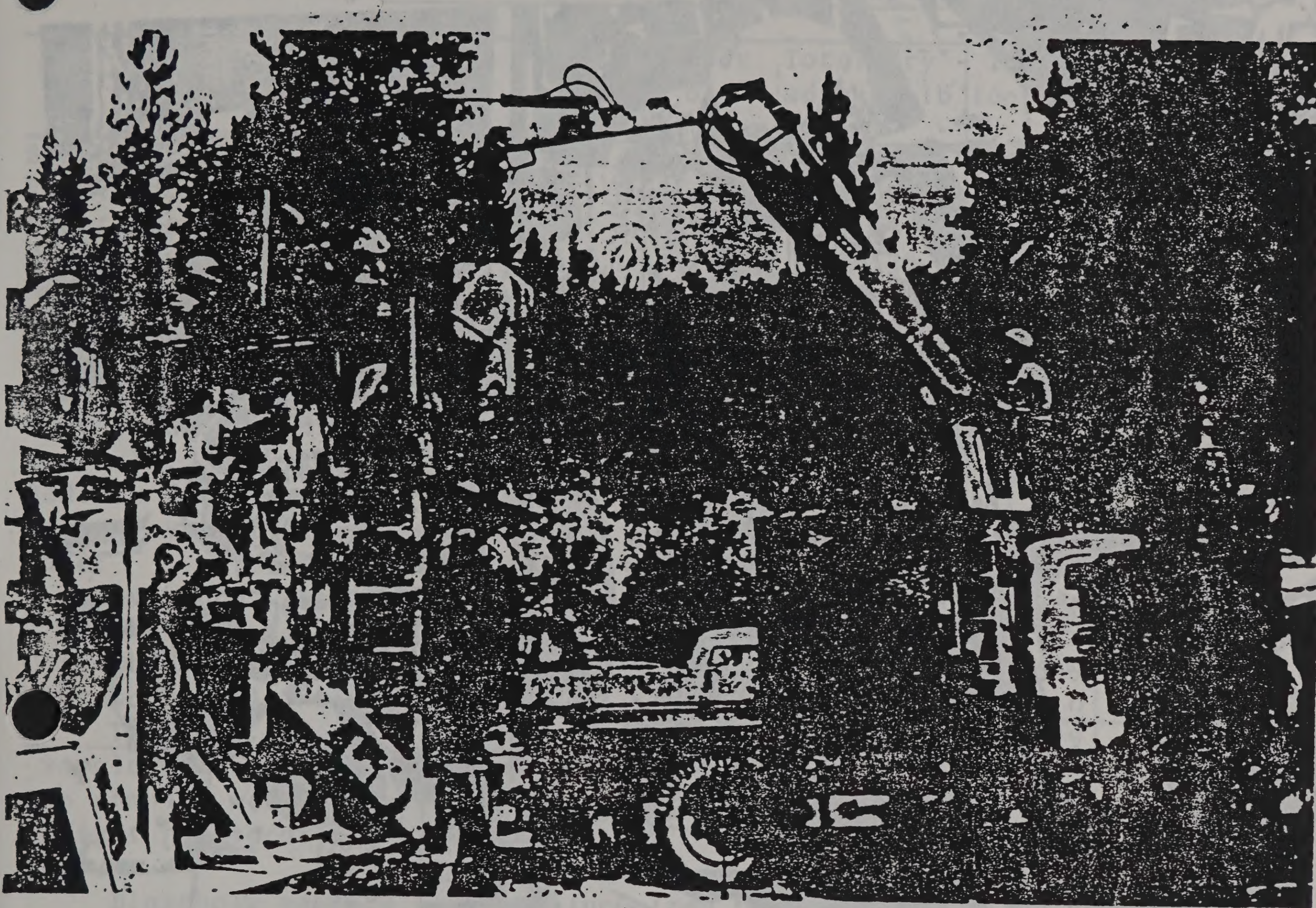

FIGURE 6. - Loading gravel into sample plant.

tandard churn drilling samples (2), which normally produce a few ounces of oncentrate and a few easily counted particles of gold.

Concentrates from Badger Hill were panned down to an average of .5 pounds per sample, from which all but the smallest particles of gold (minus 100 mesh) were removed for counting prior to amalgamation. Often large numbers of the smaller particles could not be included in the color count, thus explaining some apparent discrepancies between the color count and the total weight of the gold recovered by amalgamation. For examples of such discrepancies in the drill hole sample data, see appendix table A-4.

Another apparent discrepancy will be noted in the comparison of sample volumes and sample weights as recorded in table A-1. For a constant sample volume, the sample weights often vary considerably as a result of the relative amounts of clay, sand, gravel, and cobbles in each sample. Samples that contained a high percentage of clay and sand invariably were many pounds lighter than samples of equal volume that contained a high percentage of gravel and cobbles.

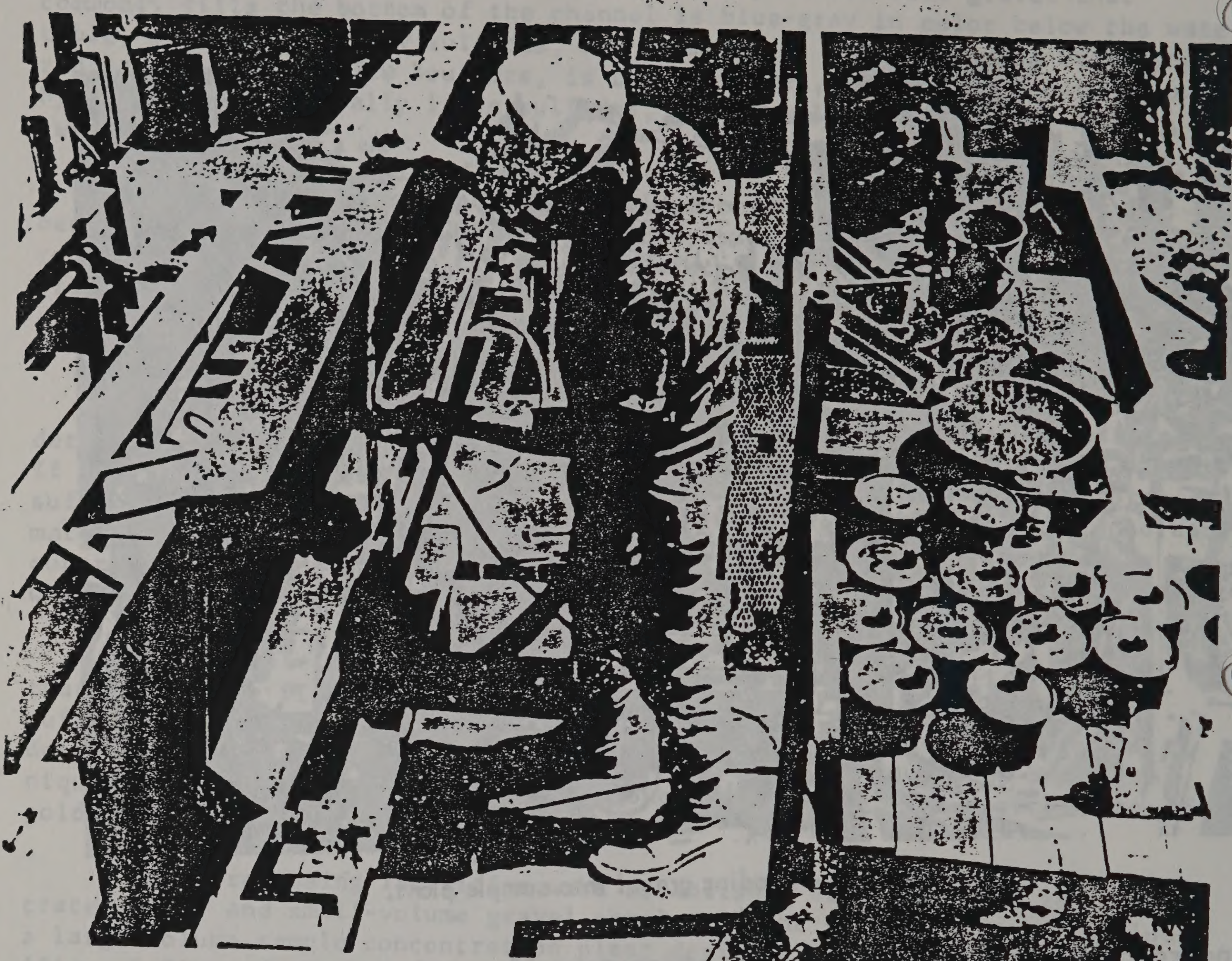

FIGURE 7. - Washing concentrate from tilting riffle box.

An automatic tailings sampler was an integral part of the concentration plant. Except for isolated instances, gold loss in the tailings was found to be negligible. Processing of tailing samples also indicated that flour gold was not adhering to the black sands and that gold was in no way chemically associated with the heavy waste materials.

Table 1 presents a resume of all types of drilling and sampling completed within the scope of the delineation studies. The drill holes and corresponding samples are classified according to type, location, or task. Bucket drill sample numbers are prefixed with the letter B and rotary drill sample numbers are prefixed with the letter C in table A-1. As indicated in table A-1, several holes were started with the bucket drill and samples were collected to the cemented gravel, from which depth a rotary drill was used to penetrate to bedrock. Hole locations are indicated in figures 3-5 (figs. 3 and 5 in pocket).

TABLE 1. - Resume of drilling and sampling at Badger Hill

Sample type	Number of holes	Number of samples	Total dry weight, lb	Total footage	Average weight per sample, lb
Bucket drill	16	160	601,744	760.9	3,760.9
Rotary drill	11	21	30,069	1,171.0	1,431.8
Blasthole	12	11	11,031	431.1	1,002.8
Hydrology hole	5	10	27,538	962.0	2,753.8
Measured pit:					
"A" series	(1)	5	13,675	(1)	2,735.0
"P" series	(1)	15	1,568	(1)	2,104.5
Stratigraphic	(1)	13	2,084	(1)	173.6
Adit round	(1)	38	91,122	(1)	2,398.0
Toe bulk	(1)	7	20,292	(1)	2,898.8
Bedrock drift (winze)	(1)	3	302	(1)	100.6
Upper level drift (raise)	(1)	9	428	(1)	47.5
Face:					
Lower bench	(1)	55	440	(1)	8.0
Lower bench select	(1)	2	1,022	(1)	811.0
Core (bedrock)	(1)	6	(3)	45.9	(3)
Vibratory drill holes	13	(2)	(2)	180.0	(2)
Total	57	355	801,315	3,550.9	2,296.0

[1] Samples were obtained by means other than drilling.
[2] No samples were collected.
[3] Weight not recorded.

Blasthole samples were obtained from two rows of surface holes (fig. 3) that were drilled in a prepared bench of blue gravel. The sample numbers correspond to the hole numbers and have a prefix of BL in table A-1.

Five hydrology test holes were rotary drilled and two samples were collected from each hole. The holes are listed as A, B, C, D, and E (fig. 3) and the corresponding samples are prefixed with the letters HH in table A-1.

One ventilation hole, VH-1 (fig. 3) was rotary drilled from the surface to the depth of the underground workings to provide additional air to the test rooms. One sample was collected and is recorded as C-14 in table A-1.

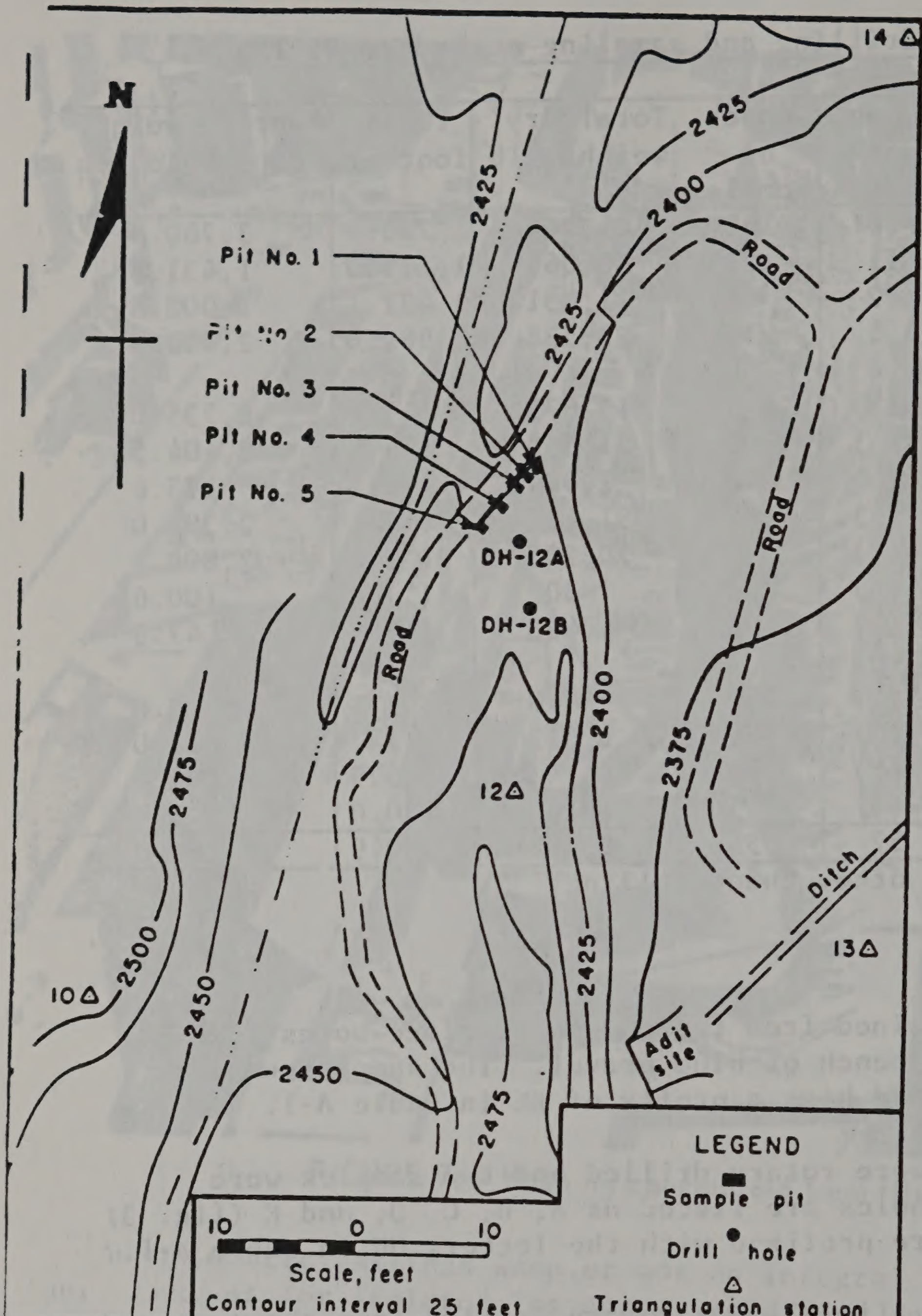

FIGURE 8. - Location of large- and small-volume pit samples and holes 12A and 12B prior to surface blasting.

Measured pit samples from pits 1-5 (fig. 8) were cut at the start of a sampling project to test statistically the reliability of large-volume samples versus small-volume samples. Early termination of the program prevented completion of this project; however, the sample results of five 1-cubic-yard samples (A samples) and 15 0.75-cubic-foot samples (P samples, fig. 9) are presented in table A-3 according to pit number and sample number.

A series of stratigraphic samples (figs. 3 and 10) were cut in a vertical face of red gravel near the edge of the lower bench. The results, prefixed with the letter 1, are given in table A-3.

Adit round samples are bulk grab samples that were collected underground after blasting each drill hole round while driving the test rooms (fig. 11). The samples are prefixed by the letters AR in table A-2.

Toe bulk samples are bulk grab samples that were obtained from the toe of each of the two masses of fragmented blue gravel produced by surface bench blasts (fig. 12). The samples are prefixed by the letters TB in table A-3.

Bedrock drift samples (fig. 11) were cut from blue gravel in the floor of the southeast-trending drift at the bottom of the winze in the Badger Hill Pit. The samples are identified in table A-2 as bedrock drift samples 1-3.

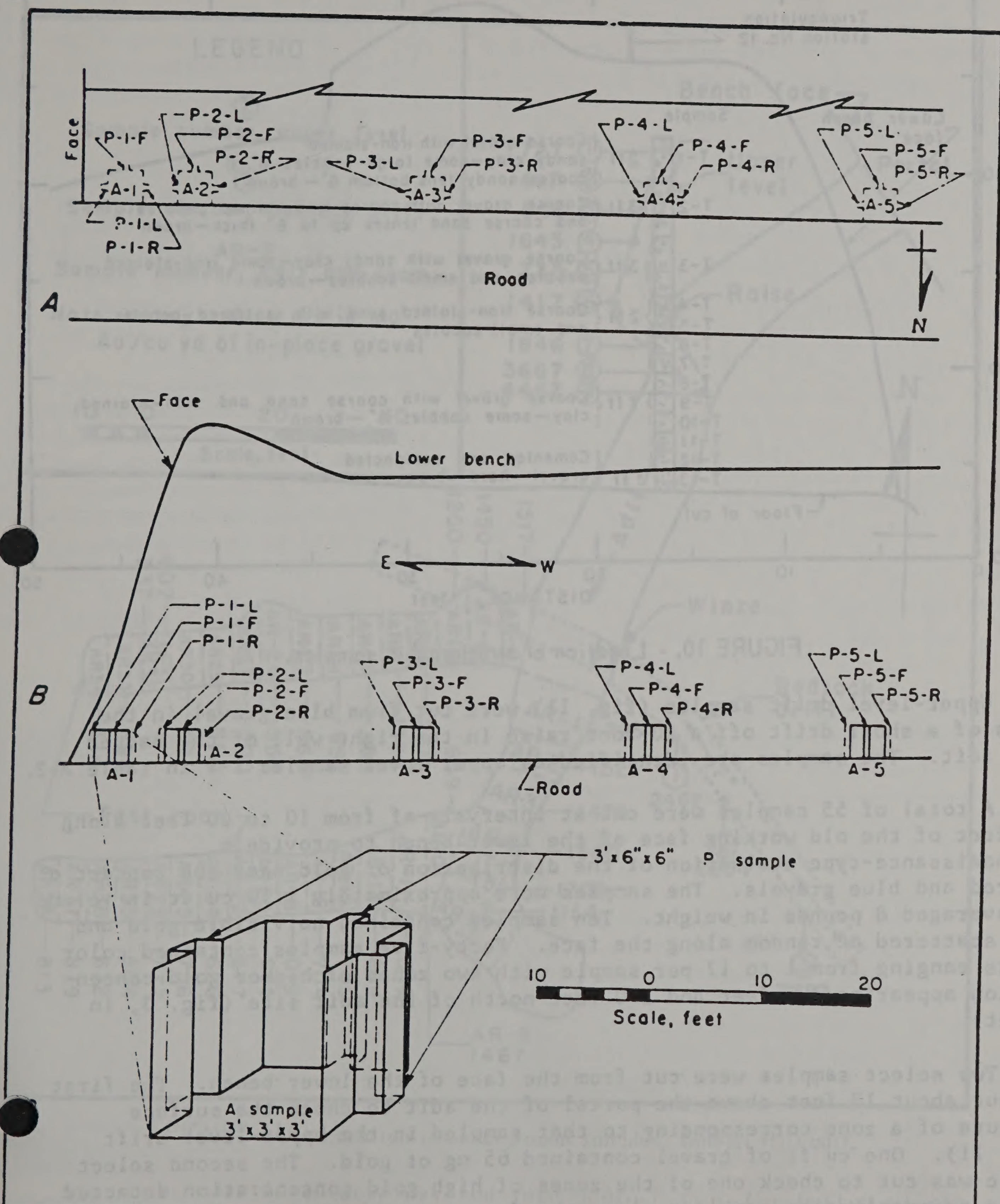

FIGURE 9. - Detail of large- and small-volume pit samples.

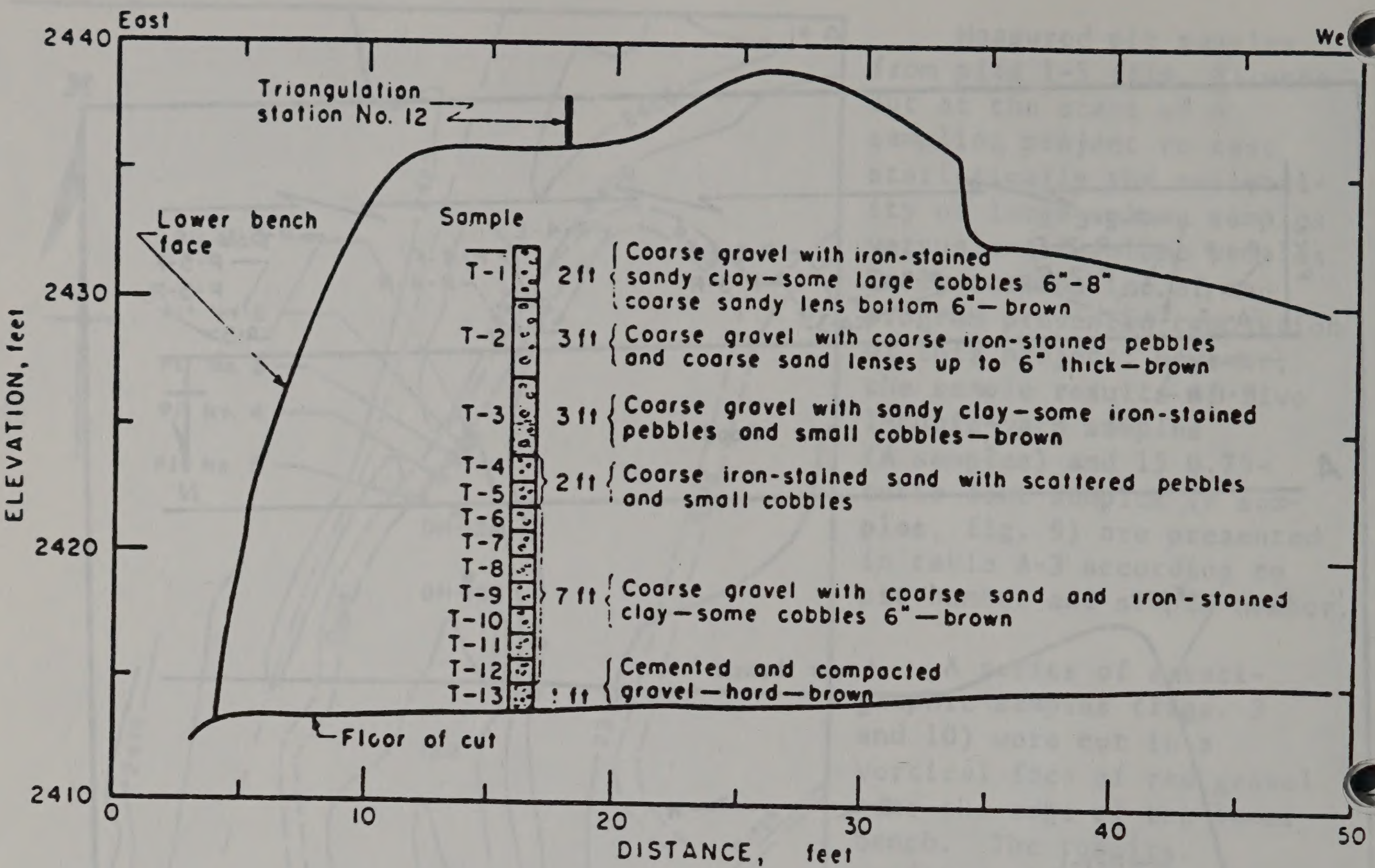

FIGURE 10. - Location of stratigraphic samples.

Upper-level drift samples (fig. 11) were cut from blue gravel in the walls of a short drift off a 20-foot raise in the right wall of the Badger Hill adit. The samples are identified as upper level samples 1-9 in table A-2.

A total of 55 samples were cut at intervals of from 10 to 20 feet along 700 feet of the old working face of the lower bench to provide a reconnaissance-type evaluation of the distribution of gold near the contact of the red and blue gravels. The samples were approximately 1/10 cu ft in volume and averaged 8 pounds in weight. Ten samples contained no visible gold and were scattered at random along the face. Forty-five samples contained color counts ranging from 1 to 17 per sample with two zones of higher gold concentration appearing 100 feet and 200 feet north of the adit site (fig. 3, in pocket).

Two select samples were cut from the face of the lower bench. The first was cut about 10 feet above the portal of the adit to check the surface exposure of a zone corresponding to that sampled in the upper-level drift (fig. 11). One cu ft of gravel contained 65 mg of gold. The second select sample was cut to check one of the zones of high gold concentration detected by the reconnaissance sampling along the face. One-third cu yd of gravel contained 45 mg of gold.

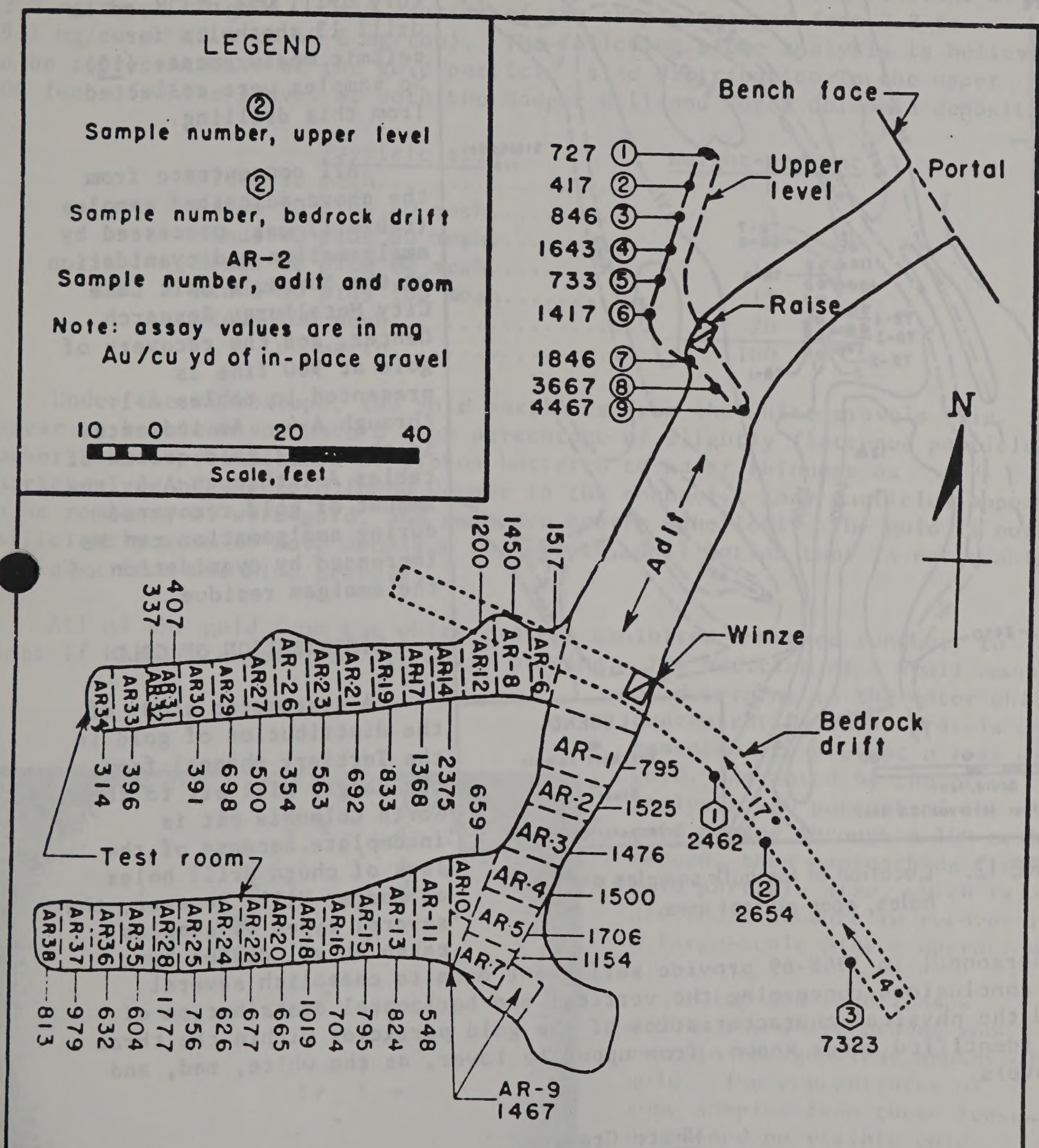

FIGURE 11. - Location of underground samples, Badger Hill adit.

Six short core holes were drilled into bedrock from the bottom of six rotary drill holes to provide core samples of rock types for the geophysics studies conducted by the Bureau of Mines (13) and the Geological Survey (4).

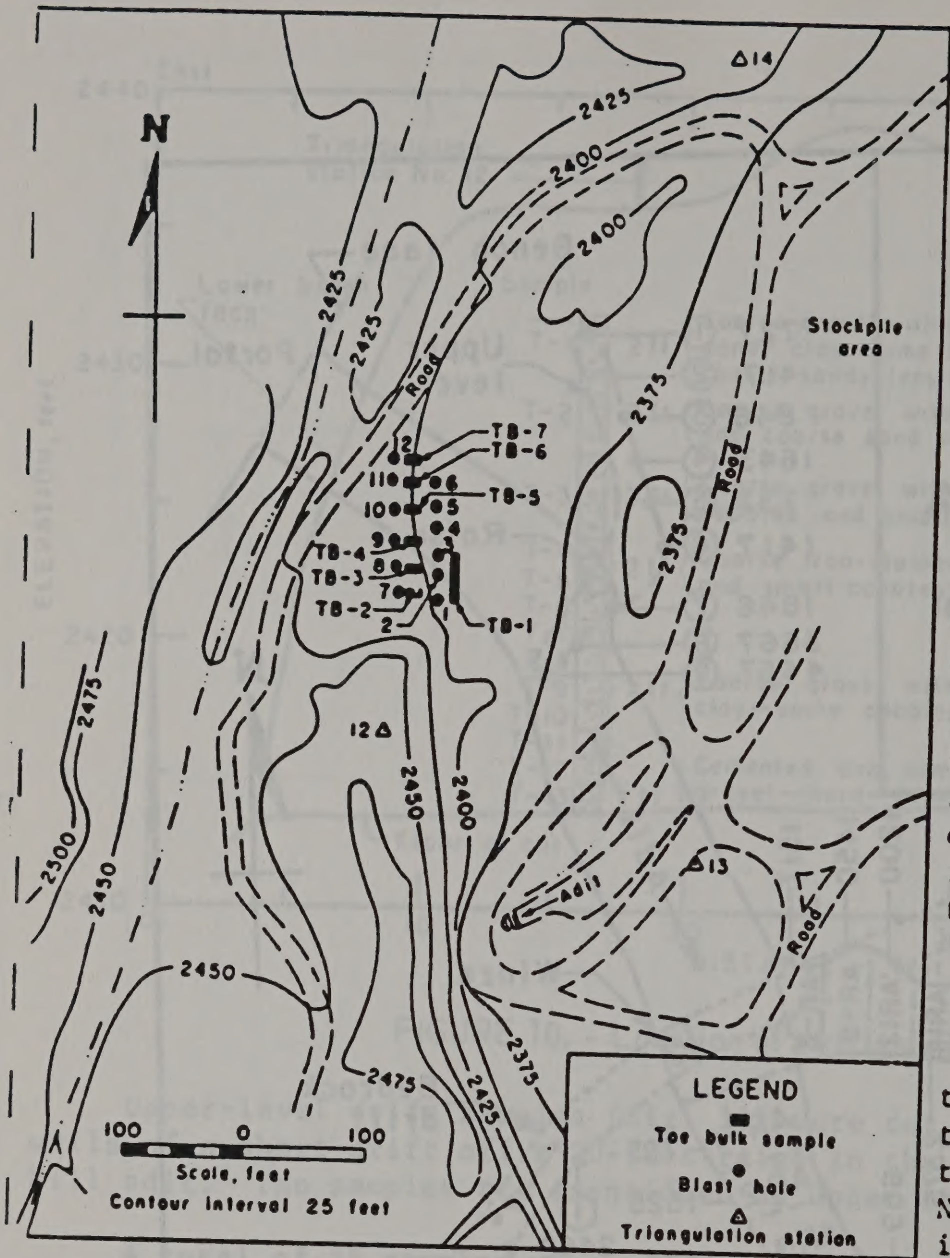

FIGURE 12. - Location of toe bulk samples and blast holes, open pit test area.

An experimental vibratory drill was utilized to drill 13 shotholes for seismic measurements (13). No samples were collected from this drilling.

All concentrate from the above-described samples (table 1) was processed by amalgamation and cyanidation at the Bureau's Salt Lake City Metallurgy Research Center, and the recovery of gold at 900 fine is presented in tables A-1 through A-4. As indicated in the total gold column of tables A-1 through A-4, the amount of gold recovered during amalgamation can be increased by cyanidation of the amalgam residue.

DISTRIBUTION OF GOLD

A systematic study of the distribution of gold in the Tertiary channel from the Badger Hill pit to the North Columbia pit is incomplete because of the lack of churn drill holes at Badger Hill; however, the several types of samples taken at all depths by Bureau personnel in 1968-69 provide sufficient data to establish several general conclusions concerning the vertical and horizontal distribution of gold and the physical characteristics of the gold particles within the three readily identified zones known, from upper to lower, as the white, red, and blue gravels.

White Gravels

An indication of the distribution and size of the gold particles found in the uppermost 90 feet of the white gravels at Badger Hill was determined from 3 samples obtained from bucket drill holes 10, 11, and 14 (fig. 3, in pocket). The samples were cut at 5-foot intervals and totaled 11,528 pounds of gravel for an average of 3,379 pounds per sample (table A-1). Twelve stratigraphic samples cut from the upper 100 feet of white gravels at the North Columbia pit

rovided a preliminary check of the gold distribution for that section of the channel.

The 111,558 pounds of upper white gravels contained only 5.5 percent of plus 1/8-inch material, mostly quartz pebbles, and contained an average of 22.2 mg/cu yd of gold. Gold content of the samples ranged from 3.2 to 9.1 mg/cu yd (2.4 to 64.8 mg/ton). The following sieve analysis is believed to be representative of the gold particle size distribution in the upper 100 feet of white gravel at both the Badger Hill and North Columbia deposits:

Particle size	Weight-percent
Minus 20 mesh..................	0
Minus 20 plus 40 mesh..........	2
Minus 40 plus 60 mesh..........	5
Minus 60 plus 80 mesh..........	11
Minus 80 plus 100 mesh.........	12
Minus 100 mesh.................	70
Composite..................	100

Under the microscope, the gold particles from the white gravels (fig. 13) ppear to be clean and have a high percentage of slightly flattened particles ather than a predominance of flakes battered to paper thinness as characterized by the gold found deeper in the channel. Many particles appear o be remnants of wire gold, and some are nearly spherical. The gold is not ufficiently worn to have acquired the smooth, dull patina that is noticeable o flakes from the blue gravels.

All of the gold from the white gravels exhibited a marked tendency to float if briefly exposed to air while panning. The addition of a small amount of detergent to the water while concentrating these gravels was essential to prevent a loss of gold. As noted by the sieve analysis, 70 percent of the gold passed through a 100-mesh screen, thus approaching flour gold particle size, which is a difficult product to recover in a large-scale placer operation.

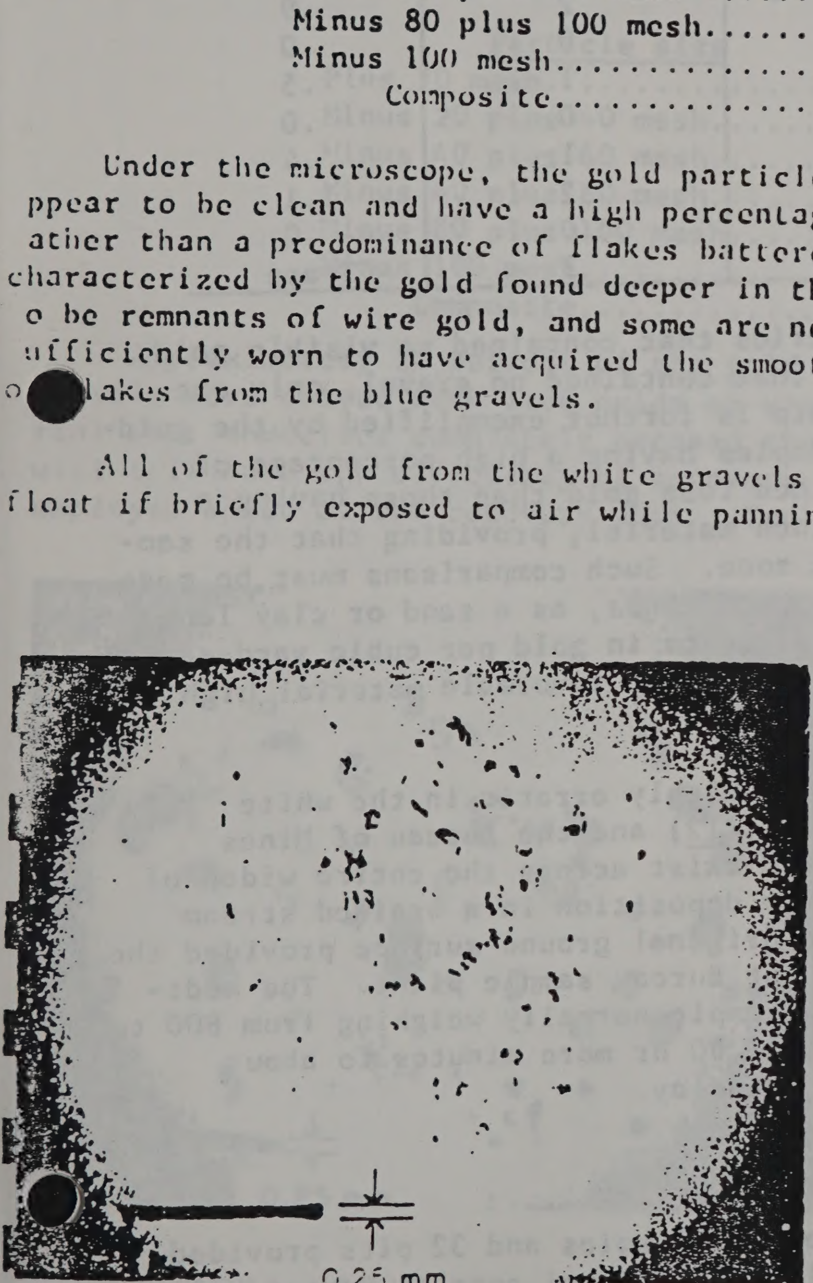

FIGURE 13. - Gold particles from white gravels, Badger Hill.

Clay and sand lenses within the white gravel zone contain the smallest amount of gold. The concentrates of some samples from these lenses contained no visible gold. This occurrence appears to be uniform throughout the deposit and is indicative of a low-velocity current from which even the smallest particles of

old normally are dropped prior to the deposition of the sand and clay. Table 2 contains analyses of twelve 10-pound samples from a vertical section of the upper white gravel zone at North Columbia, illustrating the influence of an abundance of sand and clay on the gold particle or "color" count per sample.

TABLE 2. - Analyses of 12 white gravel samples

Sample	Estimated composition, percent			Number of gold particles	Gold weight, mg
	Clay	Sand	Gravel		
1	80	20	0	0	0.0
2	70	30	0	2	Trace
3	25	70	5	6	.4
4	10	85	5	5	.3
5	15	75	10	2	.9
6	10	90	0	0	.0
7	5	80	15	1	.5
8	40	60	0	0	.0
9	10	80	10	1	.4
10	10	80	10	2	1.1
11	30	70	0	0	.0
12	40	58	2	3	Trace

Of the four samples in the above series that contained no visible gold, none contained gravel. Of five samples that contained no gravel, only one contained visible gold. This relationship is further exemplified by the gold-and-clay ratio in the bulk samples. Samples having a high percentage of minus 1/8-inch material generally contained less gold than those having a relatively low percentage of minus 1/8-inch material, providing that the samples all are from the same stratigraphic zone. Such comparisons must be made within one of the three zones and not between zones, as a sand or clay lense near bedrock possibly will contain several cents in gold per cubic yard--an amount that might equal or exceed that of the most favorable material high in the white gravel zone.

Horizontal distribution of the gold is highly erratic in the white gravels. Sampling by the San Juan Gold Co. (12) and the Bureau of Mines indicates that minor concentrations of gold exist across the entire width of the areas tested, as might be expected from deposition in a braided stream environment. A red clay zone forming the original ground surface provided the most difficult samples to concentrate in the Bureau sample plant. The addition of a measured weight of cobbles to a sample normally weighing from 800 to 1,000 pounds reduced the scrubbing time from 60 or more minutes to about [illegible] minutes to free most of the gold from the clay.

Red Gravels

A total of 192 samples from 13 bucket drill holes and 32 pits provided [illegible]08,143 pounds of gravel from the central or red gravel zone at Badger Hill. The bucket drill holes were spotted in the floor of the middle bench (fig. 3, in pocket) and were extended to the contact of the highly compacted blue

gravel that usually stopped the drilling advance after a few inches of penetration. A perched water table rests upon the contact of the red and blue gravels. The thickness of the red gravel zone ranges from 20 to 80 feet in the area tested.

Gold particle size and the amount of gold per cubic yard increased noticeably from the white to the red gravels, although the changes probably are gradational. In the white gravel zone, bucket drill hole samples yielded an average of 22.2 mg/ton of gold (30 mg/cu yd) at 900 fine; and in the red gravel zone, 490,816 pounds of bucket drill hole samples yielded an average of 43.5 mg/ton (58.8 mg/cu yd) at 900 fine. Plus 1/8-inch material increased from 5.5 percent in the white gravel to 46 percent in the red gravel.

The following sieve analysis is believed to be representative of the gold particle size distribution for the red gravels at Badger Hill:

Particle size	Weight-percent
Plus 20 mesh......................	0
Minus 20 plus 40 mesh...........	36
Minus 40 plus 60 mesh...........	38
Minus 60 plus 80 mesh...........	17
Minus 80 plus 100 mesh..........	5
Minus 100 mesh..................	4
Composite..................	100

Approximately 20 percent of the gold particles observed from the red gravels were coated with iron oxide or with an unidentified translucent material that sometimes completely encased the flakes. Both types of coatings will influence the efficiency of recovery by amalgamation unless scrubbing is employed ahead of the recovery unit. Ten minutes of scrubbing in the Bureau sample plant normally was sufficient to remove the coating and to brighten most of the rusty gold. Much of the gold will float readily if allowed to become dry or greasy at any stage of the recovery operation.

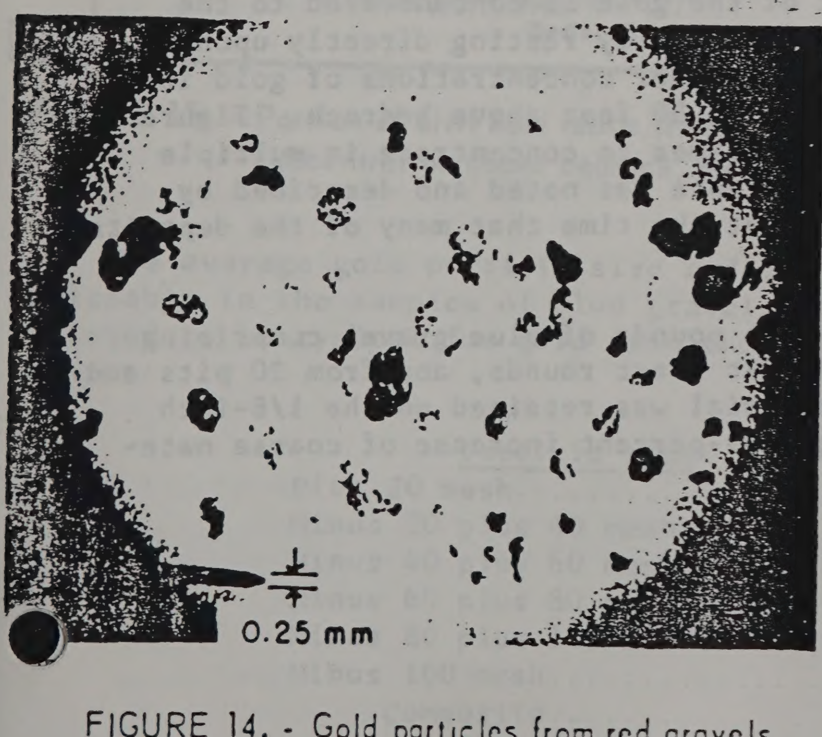

FIGURE 14. - Gold particles from red gravels, Badger Hill.

As indicated by the sieve analysis, the gold flakes generally are larger in size than those from the white gravels and are characterized by a thin, flat, pancake appearance (fig. 14). A few rough and sometimes spherical particles were observed in most of the samples, possibly indicating more recent liberation and less battering through transportation.

Sample data (table A-1) indicate that no concentration of gold exists at the contact of the unconsolidated red gravels and the highly compacted or cemented blue gravels. Although the transition from one zone to the other often is sharply defined, the contact apparently did not materialize in time to serve as a false bedrock.

As noted in the white gravels, the least favorable zones for the concentration of gold are those having a high content of clay and sand. The stratigraphic section (fig. 10, table A-3) illustrates the selective occurrence of gold in several feet of red gravel, sand, and clay. The horizontal distribution also appears to be similar to that in the white gravels whereby minor concentrations of gold were found across the full width of the areas tested.

Blue Gravels

Unoxidized or blue gravels cover the deepest portion of the channel to known depths ranging from 118 to 138 feet along the trough at Badger Hill and from 100 to 250 feet along the trough at North Columbia. This zone is characterized by a distinctive gray-blue color, high density, and a relatively high percentage of large boulders within 20 feet of bedrock. The material is highly compacted and usually is tightly cemented, requiring drilling and blasting for primary fragmentation.

Approximately 80 percent of the total gold content of the channel deposit is believed to be confined to the blue gravels. Sampling data developed by the Bureau of Mines tend to confirm the earlier work at the North Columbia deposit (12), which indicate that most of the gold is concentrated to the lower 40 feet of blue gravels, but not necessarily resting directly upon bedrock. Approximately 50 percent of the richer concentrations of gold in the tested areas at North Columbia are from 5 to 30 feet above bedrock. Figure 15 illustrates the tendency for the gold sometimes to concentrate in multiple lenses above bedrock. This type of occurrence was noted and described by Lindgren (8, p. 66) and Munro (9, p. 28) at the time that many of the deposits were being mined.

The Bureau of Mines collected 154,266 pounds of blue gravel comprising 92 samples from 33 drill holes, from 38 adit blast rounds, and from 20 pits and cuts. Seventy-seven percent of this material was retained on the 1/8-inch screen of the sample plant, indicating a 31-percent increase of coarse material over that of the red gravels.

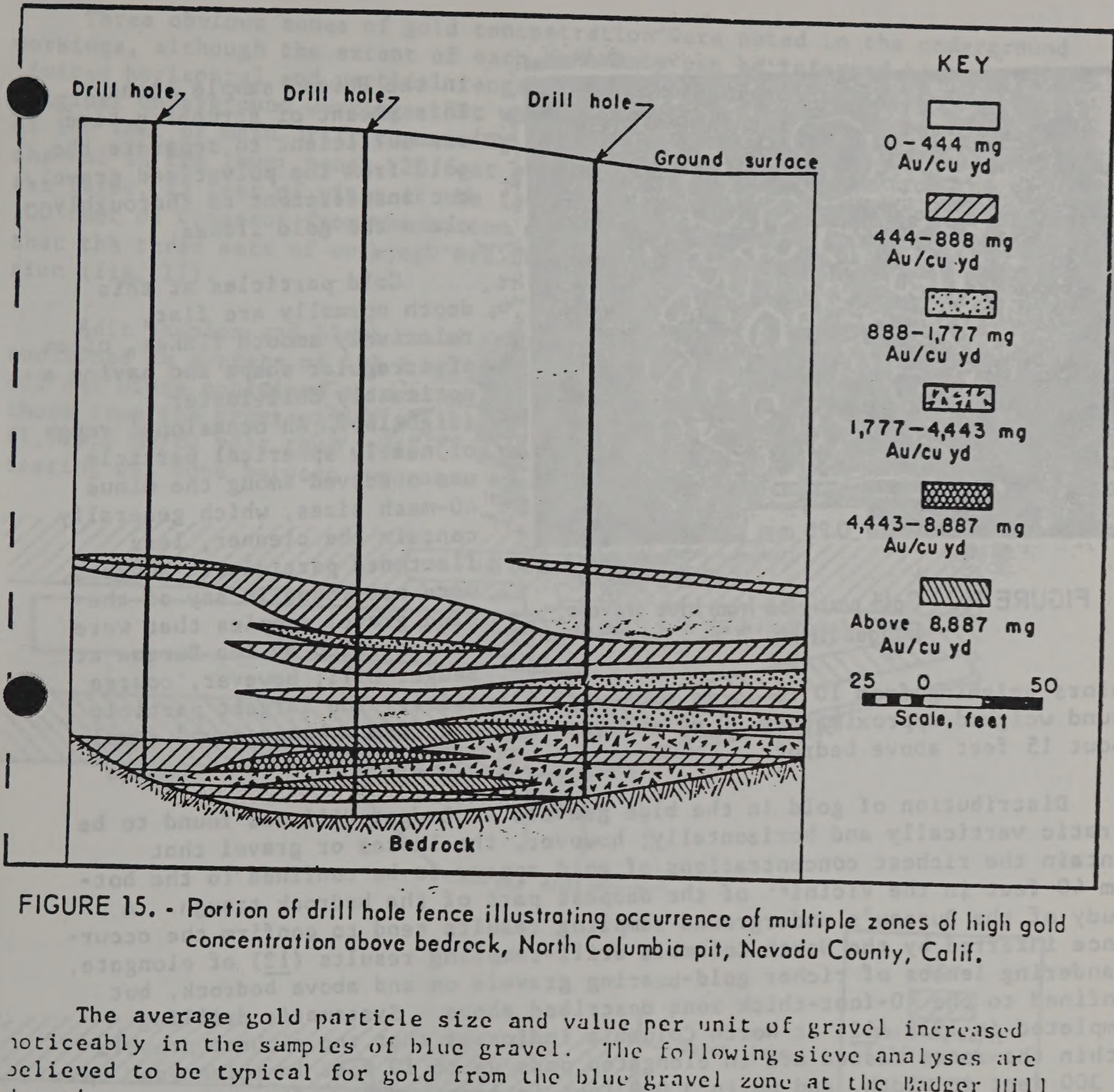

FIGURE 15. - Portion of drill hole fence illustrating occurrence of multiple zones of high gold concentration above bedrock, North Columbia pit, Nevada County, Calif.

The average gold particle size and value per unit of gravel increased noticeably in the samples of blue gravel. The following sieve analyses are believed to be typical for gold from the blue gravel zone at the Badger Hill deposit:

Particle size	Weight-percent
Plus 20 mesh	24.7
Minus 20 plus 40 mesh	48.2
Minus 40 plus 60 mesh	19.2
Minus 60 plus 80 mesh	7.7
Minus 80 plus 100 mesh	.2
Minus 100 mesh	.0
Composite	100.0

Approximately 50 percent of the gold particles still were coated with varying amounts of blue-black material after from 5 to 10 minutes of scrubbing

FIGURE 16. - Gold particles from blue gravels, Badger Hill.

in the Bureau sample plant. This amount of scrubbing time was sufficient to separate the gold from the pulverized gravel, but insufficient to thoroughly clean the gold flakes.

Gold particles at this depth normally are flat, relatively smooth flakes, often of irregular shape and having a noticeably dull luster (fig. 16). An occasional rough or nearly spherical particle was observed among the minus 40-mesh sizes, which generally contain the cleaner, less flattened particles. Nuggets were not found in any of the blue gravel samples that were concentrated by the Bureau at Badger Hill; however, coarse colors weighing from 10 to 15 mg each were plentiful. The largest particle found weighed approximately 30 mg and was obtained from a blast round sample about 15 feet above bedrock.

Distribution of gold in the blue gravels at Badger Hill was found to be erratic vertically and horizontally; however, the lenses or gravel that contain the richest concentrations of gold appear to be confined to the bottom 40 feet in the vicinity of the deepest part of the bedrock trough. A study of the Bureau's underground sampling results tend to confirm the occurrence inferred by the North Columbia drill-sampling results (12) of elongate, meandering lenses of richer gold-bearing gravels on and above bedrock, but confined to the 40-foot-thick zone described above. Systematic drilling completed in 1938 (12) at North Columbia indicates that the richest gravels within the zones tested are in elongated pods up to 10 feet in thickness, up to 300 feet in width, and up to 3,000 feet in length that reflect the meandering course of the deepest parts of the channel.

The Bureau of Mines drill-sampling project was too limited to determine the approximate length and width of the richer concentrations of gold at Badger Hill; however, the underground sampling confirmed the existence of multiple zones on and above bedrock.

Fifty samples totaling 91,852 pounds of blue gravel were collected from the underground workings in the lower bench. They contained gold ranging in value from 177.7 to 3,752.0 mg/ton (313.7 to 7,322.6 mg/cu yd) and averaging 431.2 mg/ton, or 761.8 mg/cu yd. Collection points ranged from a few inches to 25 feet above bedrock. Table A-2 gives winze, raise, and adit blast round sample data, and figure 11 gives the sample location map.

Three obvious zones of gold concentration were noted in the underground workings, although the extent of each zone only can be inferred because of the limited horizontal and vertical range of the crosscuts and drifts. The original underground workings that were rehabilitated by the Bureau consisted of 100 feet of main drift commencing on bedrock at the deepest part of the channel in the lower bench, 20 feet of raise, 42 feet crosscut from the top of the raise, 15 feet of winze at the face of the main drift, and approximately 100 feet of crosscut from the bottom of the winze. Sample results indicate that the three sets of workings are in three different zones of gold deposition (fig. 17).

Adit samples and blast round samples collected from the test rooms contained an average of 422.2 mg/ton (747.0 mg/cu yd); those from the crosscut off the winze contained an average of 2,145.7 mg/ton (4,337.3 mg/cu yd); and those from the crosscut off the raise contained an average of 981.3 mg/ton (1,779.6 mg/cu yd); thus indicating that most of the Bureau's underground testing was done between two zones of higher grade gold-bearing gravel.

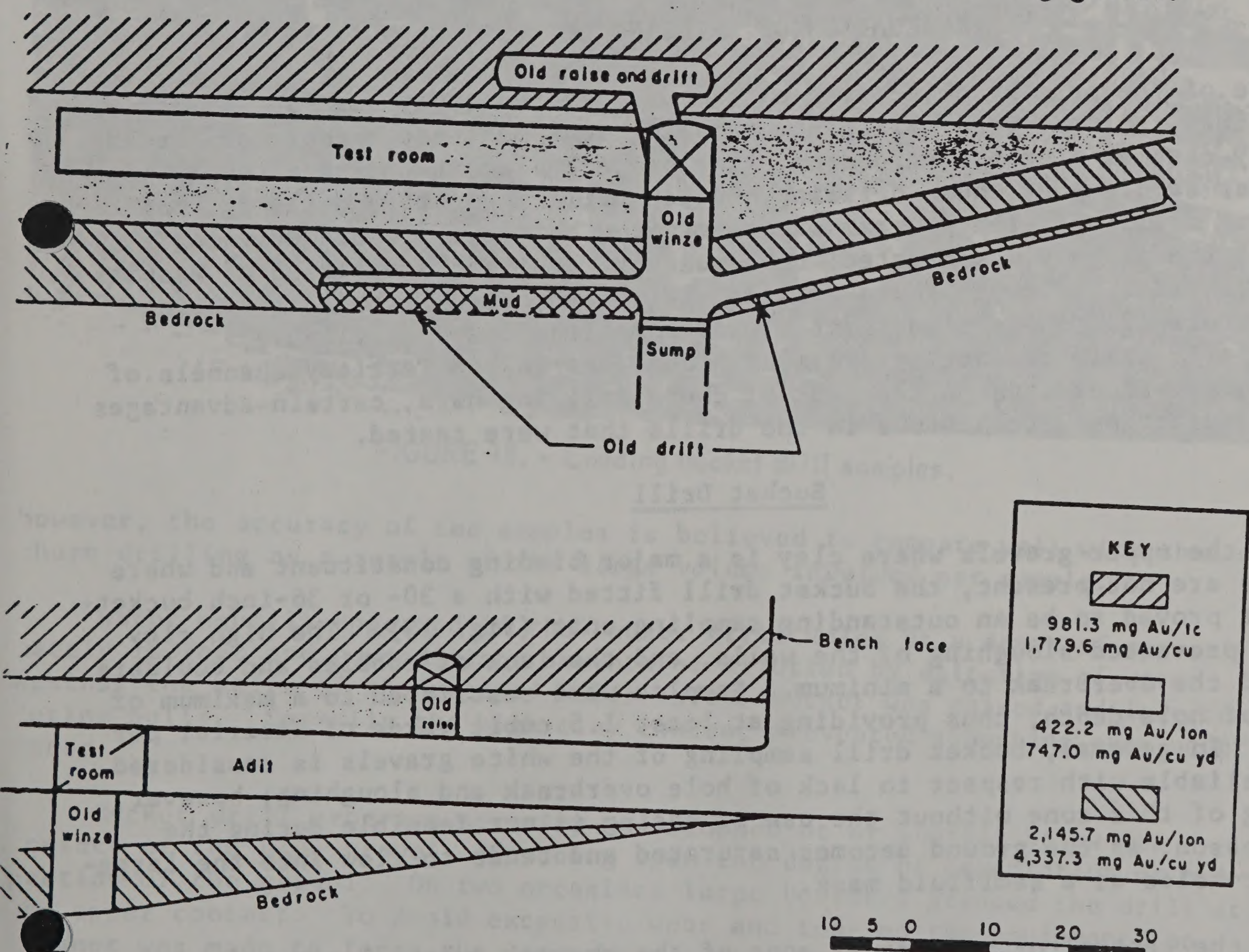

FIGURE 17. - Cross section (top) and longitudinal section (bottom) through Badger Hill adit showing zones of gold-bearing gravel.

Cemented gravel resting on bedrock at the portal of the adit contained little more than a trace of gold from repeated tests; however, samples of bedrock gravels taken from the crosscut off the winze about 100 feet in from the portal contained gold values ranging from 1,243 to 3,752 mg/ton (2,461 to 7,323 mg/cu yd). As indicated on the sample location map (fig. 11), the richest sample, containing 7,323 mg/cu yd, is up slope to a point where it is at the same elevation as the floor of the adit, and the lowest grade sample is at the deepest end of the crosscut.

The horizontal extent of the highest concentrations of gold in the blue gravels was not determined at Badger Hill; however, 31,528 pounds of blue gravel obtained from 11 drill holes and 7 pits ranging in distance from 250 to 400 feet from the underground workings contained an average of 186.3 mg/ton or 340.1 mg/cu yd. This is a drop of 244.9 mg/ton or 421.7 mg/cu yd from the underground sample average, indicating that the gold content of the blue gravels decreases sharply as sampling progresses away from the deepest portion of the bedrock trough.

EVALUATION OF DRILLING METHODS

One of the primary objectives of the delineation study was to test and, if possible, to improve drilling and sampling equipment and techniques for the Tertiary channel placer environment. The project plans required a comparison of placer sampling by churn drilling, bucket drilling, rotary drilling, and vibratory drilling. The program was terminated before the churn drilling phase of the project was started; consequently, a direct comparison of results cannot be made. Although churn drilling is the accepted method of sampling deep placers (2), it is slow, costly, and sometimes not too reliable, particularly where cemented gravels are found, as in the Tertiary channels of California. Regardless of the lack of churn drilling data, certain advantages and disadvantages were obvious in the drills that were tested.

Bucket Drill

In the upper gravels where clay is a major binding constituent and where boulders are not present, the bucket drill fitted with a 30- or 36-inch bucket-type bit proved to be an outstanding sampling unit (fig. 18). The high clay content prevented sloughing of the walls, and the lack of cobbles and boulders lessened the overbreak to a minimum. Samples were restricted to a maximum of 5 feet of hole depth, thus providing at least 1.5 cubic yards of material per sample. In general, bucket drill sampling of the white gravels is considered to be reliable with respect to lack of hole overbreak and sloughing; however, drilling of this zone without the use of casing is not feasible during the rainy season, as the ground becomes saturated and tends to flow into the large-diameter holes as a semifluid mass.

In the central or red gravel zone of the channel, where lenses of loose sand and gravel up to several feet in thickness occur, 32-inch casing was forced through the thicker sections of loose material using high drillhead pressure. Some dilution was noted in this zone caused by sloughing of the thinner lenses of loose gravel and by tearing of the walls by cobbles;

FIGURE 18. - Loading bucket drill samples.

however, the accuracy of the samples is believed to compare well with that of churn drilling as a result of the large volume involved per sample.

Water usually appeared in the bucket drill holes at a depth of approximately 18 feet and caused minor amounts of dilution by agitation of water against the walls of the uncased holes as the bucket was raised and lowered during pulls. Normally the high clay content prevented sloughing even under water.

Bucket drill progress normally was stopped at or shortly below the contact of the blue gravel, depending upon the degree of cementation or compaction of the gravel. On two occasions large boulders stopped the drill at the upper contact. To avoid excessive wear and tear on the equipment, no attempt was made to force the penetration into the blue gravels.

In general, the bucket drill proved to be an outstanding large-volume sampling tool for the white and red gravels. It has little or no application in the highly compacted and cemented blue gravels. If sample volume and time

are essential factors, the bucket drill has an advantage over the churn drill; however, it is not able to penetrate the cemented gravel and it requires the availability of large-scale processing equipment to concentrate the high-volume samples.

Rotary Drill

A standard, truck-mounted rotary drill (fig. 19) used by the Bureau and a second similar drill used by a contractor for the hydrology study were of greatest value in drilling rapidly to bedrock through the cemented blue gravels. The delineation holes were drilled using mud as a circulation fluid. The hydrology holes were drilled using large volumes of compressed air to remove the saturated cuttings. Compressed air would not remove thick mud, but with the addition of sufficient water to produce a loose mixture, both the cuttings and water were blown from the hole as a slurry.

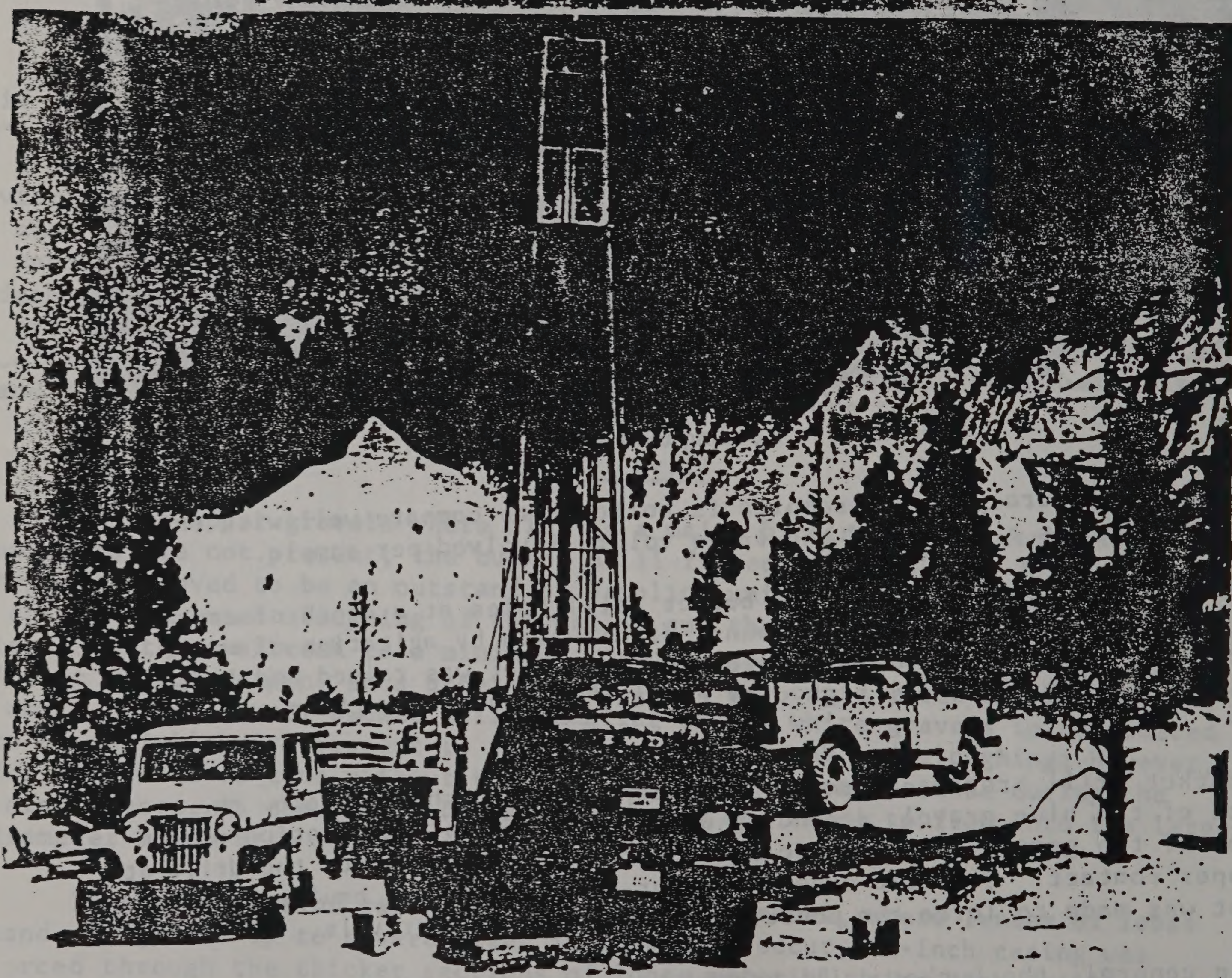

FIGURE 19. - Rotary drilling in cemented gravel.

Casing was not required to hold the walls of most of the holes that were drilled in the blue gravel; however, one hole was lost as a result of penetrating a zone of loose sand near bedrock. With the use of mud or air as a circulating agent to remove cuttings, the value of the rotary drill as a relatively precise sampling tool for the Tertiary channel gravels is nil. The use of heavy mud and slow penetration rates did not prevent excessive dilution of the samples. Water alone or water and compressed air used as circulating agents tend to produce more dilution than occurs when the holes are coated with drilling mud.

Because of the high dilution factor, which often approached and sometimes exceeded 100 percent, no attempt was made to sample the holes at closely spaced intervals. Cuttings were collected as a single sample from the upper contact of the blue gravels to an estimated depth of 20 feet above bedrock. A second sample was collected from this point to bedrock. The two samples were collected primarily to determine whether or not a sharp increase of gold occurs from the upper to the lower zones of the blue gravel section and to obtain a full thickness sample of gold from the blue gravel.

With additional experimental work utilizing casing and short runs, a fairly reliable sampling procedure possibly could be devised for rotary drilling in the cemented gravels. Under this program, the rotary drill's principal value was found to be in providing rapid penetration to bedrock for a specific purpose, such as verification of geophysical work or to provide holes for hydrology studies. One hole was drilled quickly from the surface to the level of the bedrock drift for ventilation purposes.

Vibratory Drill

The vibratory drilling concept has been utilized successfully in the construction business for driving heavy steel piling using high-frequency vibrations transmitted through the piling to literally fluidize the surrounding soil and allow the piling to sink rapidly to the desired depth. A truck-mounted experimental drill (fig. 20) using the vibratory concept was tested by the Bureau in many types of material to determine the potential of the drill for capturing samples; however, the result was persistent and almost immediate blocking of the core barrel. Extensive experimental work was performed in an effort to overcome this deficiency, but the Heavy Metals Program was terminated before the work was completed.

At Badger Hill, the vibratory drill performed exceptionally well in the blue and red gravels to drill seismic shotholes to required depths ranging from 10 to 15 feet. A drilling time of from 1 to 2 minutes per hole was adequate for these depths. Although clay immediately blocked the drill pipe, the penetration rate was not slowed. Until a core barrel or tube is designed that will allow the clay and gravel to enter without blocking, the drill will be of little utility for sampling purposes in the Tertiary channels.

FIGURE 20. - Vibratory drill placing shotholes.

DISCUSSION OF DELINEATION COSTS

Drilling and sampling costs incurred by the Bureau of Mines at Badger Hill have little direct application to the estimation of actual delineation costs because of the research orientation of the Bureau activities. As no tool was tested or developed by the Bureau that is superior to the churn drill for penetrating and sampling the blue gravels, this type of drill is expected to be utilized at least in part by any company that might consider conducting additional delineation work in the channel.

The most recently completed churn drilling on San Juan Ridge was contracted by the U.S. Geological Survey in the summer of 1968 at a cost of approximately $25 per foot. Considering the scarcity of experienced gold placer churn drillers, even a relatively large drilling contract probably would fall into the range of at least $25 per foot as of yearend 1973.

Realistic cost estimates for additional delineation on San Juan Ridge will be controlled by the system of mining that is proposed. If the entire deposit is to be mined for various byproducts as well as for gold

Sampling of at least two types will be required to investigate the entire thickness of the channel fill to supplement the available sampling data and to generate new data concerning the amount and distribution of the potential byproducts. Ilmenite, zircon, chromite, garnet, and rutile were identified in the concentrate from Badger Hill. Trench sampling of the old working faces can be accomplished quickly and economically to determine the amount and distribution of the byproduct materials in the white gravels. Trench sampling sites are accessible at the Badger Hill, Cherokee, North Columbia, and Spring Creek pits that would provide adequate sample control for the entire 6-mile segment of lava-free channel.

Approximately 50 channel samples averaging 40 cubic yards per sample strategically spotted among the old pits should provide reasonably accurate data with regard to the amount and distribution of the potential byproducts. The cost of a sampling program as of 1971, including equipment, labor, and operating costs, will range from $45,000 to $50,000 for a period of from 4 to 5 months on a 3-shift basis during the dry season. Most of this time will be spent in setting up, in gaining access to sample sites, and in the actual digging of the samples. To process approximately 2,000 cubic yards of gravel with a relatively small and simple trommel-type concentration unit will require little more than 30 operating days.

Regardless of the decision to process or to waste the white and red gravels, additional delineation data will be required concerning the position and extent of the deepest and most favorable portion of the blue gravels. On the basis of drill-sampling data presently available, this zone is expected to be confined to a band seldom exceeding 500 feet in width and 40 feet in thickness that follows the configuration and meanders of the deepest portion of the bedrock trough. Drill sampling should be confined to this zone as closely as possible to reduce the high cost of penetrating and recovering samples in the tightly compacted and cemented blue gravels.

The deepest portion of the channel can be determined by geophysical techniques to reduce the target area to a minimum prior to drilling (10, 13). With the proper and successful utilization of geophysics, the fences of drill holes can be reduced to three or four holes in possibly five fences spotted along the least-explored portion of the channel from the North Columbia pit to the lava capping about 2 miles to the east. An additional three to four fences might be required between the Badger Hill and North Columbia pits to provide more detailed data than presently is available from the old records.

The cost of preliminary delineation (as of 1971) of the deepest portion of the trough along the entire channel is estimated to be $25,000. Such a study would utilize seismic and gravity methods and would require three men for a period of from 5 to 6 months including interpretation time. A drilling program based upon the geophysics data and ranging from 24 to 26 holes averaging 300 feet in depth will cost from $110,000 to $160,000 including access.

CONCLUSIONS

Bureau of Mines sample results from the Badger Hill depo: supplemented by invaluable drilling and sampling data provided by the San Juan Gold Co. (5, 7, 9, 11-12) indicate that the gravels containing the greatest concentrations of gold are confined to the blue zone within 40 feet of bedrock in the deepest part of the channel. The width of the most favorable zone is estimated to vary from 100 to 500 feet; however, the actual width, thickness, and direction of the zone will be determined by the configuration, slope, and meanders of the original bedrock trough. The richest concentrations of gold within the zone are expected to occur in isolated, elongated lenses of gravel that may or may not be on bedrock. Two or more rich lenses may occur one above the other from 5 to 40 feet above bedrock, creating additional exploration and mining problems for maximum recovery by underground methods. A rapid, inexpensive drill-sampling technique will be essential for underground use to probe for multiple lenses of rich gravel unless a full mining height of 40 feet is maintained along bedrock.

Gold was found to be universally distributed in irregular amounts throughout the white and red zones. The quantity and particle size of the gold increased progressively with depth in all three zones. Lenses of sand and clay contained the least amount of gold regardless of location.

None of the three drills that were tested proved to be an all-purpose tool that can compete with the standard placer-type churn drill. Each of the three drills has a limited use for special conditions and each in its restricted capacity provides satisfactory results; however, the data provided by earlier drilling records (12) indicate that in the blue gravel the churn drilling technique continues to have superior capability for providing significantly reliable samples. Additional work is needed towards the development of a cheaper, quicker, and more reliable method to drill and sample the cemented gravel, as even the reliability of the churn drill sample is greatly lowered when the casing cannot be driven ahead of the bit.

In providing drilling support for the geophysics studies, a standard, truck-mounted rotary drill was found to be essential to quickly penetrate the cemented and highly compacted gravels to precisely identify the depth of bedrock.

Large-volume samples were difficult to handle and required specialized equipment for processing, but the relatively large quantities of gold and concentrate that were recovered from each sample provided additional reliability for the determination of the quantity and size distribution of the gold.

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APPENDIX.--DRILLING AND SAMPLING DATA

TABLE A-1. - Drill hole sample data

Hole	Diameter, inches	Sample	Sample interval, feet	Net weight, pounds	Dry weight, pounds	Plus 1/8-inch fraction, percent	Approximate volume, cubic yards	Gold recovery, milligrams	Total gold,[1] milligrams	Type hole
1....	36	B-1	2 - 7	6,333	5,700	51.2	1.68	72.50	82.65	Bucket.
	36	B-2	7 - 12	6,127	5,514	49.0	1.68	85.64	91.34	Do.
	36	B-3	12 - 17	5,420	4,878	49.2	1.68	63.43	66.30	Do.
	36	B-4	17 - 22	7,096	6,244	60.1	1.68	47.87	51.18	Do.
	36	B-5	22 - 27	8,131	7,074	64.6	1.68	113.73	120.54	Do.
	36	B-6	27 - 32	6,589	5,798	46.7	1.68	121.19	127.10	Do.
	36	B-7	32 - 34.7	3,953	3,479	50.7	.91	Sample	Lost	Do.
	5	C-10	34.7-109.3	1,516	1,410	38.1	.38	57.00	61.40	Rotary.
	5	C-11	109.3-129.3	514	478	12.8	.10	4.50	4.50	Do.
2....	36	B-8	2 - 5	6,309	5,867	51.2	1.01	265.04	284.04	Bucket.
	36	B-9	5 - 10	8,867	7,803	50.2	1.68	265.10	282.01	Do.
	36	B-10	10 - 15	6,411	5,770	57.5	1.68	79.69	86.34	Do.
	36	B-11	15 - 20	6,533	5,880	58.7	1.68	136.15	151.57	Do.
	36	B-12	20 - 25	7,651	6,733	57.4	1.68	251.09	276.36	Do.
	36	B-13	25 - 25.4	635	559	68.2	.13	43.71	44.46	Do.
3....	36	B-14	0 - 5	5,609	5,048	47.6	1.68	184.79	196.28	Do.
	36	B-15	5 - 10	5,329	4,690	62.7	1.68	17.16	18.72	Do.
	36	B-16	10 - 15.2	4,838	4,354	67.5	1.75	62.35	66.60	Do.
	5	C-3	15.2-106.2	2,760	2,567	36.1	.46	200.00	206.80	Rotary.
4....	36	B-17	0 - 5	5,868	5,457	42.1	1.68	22.40	22.60	Bucket.
	36	B-18	5 - 10	5,984	5,565	43.6	1.68	222.15	227.83	Do.
	36	B-19	10 - 15	5,776	5,372	44.0	1.68	508.38	527.84	Do.
	36	B-20	15 - 20	5,640	5,189	47.6	1.68	121.37	124.74	Do.
	36	B-21	20 - 25	6,186	5,567	47.3	1.68	112.49	120.56	Do
	36	B-22	25 - 30	7,876	6,931	48.3	1.68	128.74	133.53	Do.
	36	B-23	30 - 35	6,894	6,205	48.7	1.68	121.35	126.28	Do.
	36	B-24	35 - 40	7,713	6,787	60.4	1.68	153.40	156.40	Do.
	36	B-25	40 - 41.4	2,102	1,829	63.6	.47	57.82	61.62	Do.
5....	36	B-26	0 - 5	5,888	5,476	47.9	1.68	306.92	319.66	Do.
	36	B-27	5 - 10	6,421	5,650	59.4	1.68	60.50	73.10	Do.
	36	B-28	10 - 15	6,141	5,527	56.8	1.68	45.50	70.90	Do.
	36	B-29	15 - 20	6,649	5,984	63.5	1.68	43.50	62.50	Do.
	36	B-30	20 - 20.3	645	568	62.8	.10	8.00	8.40	Do.
	5	C-1	22 -112	2,799	2,603	33.0	.46	264.00	268.50	Rotary.
	5	C-2	112 -133	536	498	23.9	.11	185.00	189.30	Do.
[illegible]....	36	B-31	0 - 5	5,789	5,210	60.3	1.68	95.50	112.80	Bucket.
	36	B-32	5 - 10	7,039	6,194	53.7	1.68	230.00	347.44	Do.
	36	B-33	10 - 12	972	875	70.2	.67	28.00	41.11	Do.
	5	C-4	12 -122	3,257	3,029	36.4	.56	177.50	183.30	Rotary.
	5	C-5	122 -142	744	692	31.7	.10	200.00	234.60	Do.

See footnotes at end of table.

TABLE A-1. - Drill hole sample data--Continued

Hole	Diameter, inches	Sample	Sample interval, feet	Net weight, pounds	Dry weight, pounds	Plus 1/8-inch fraction, percent	Approximate volume, cubic yards	Gold recovery, milligrams	Total gold,[1] milligrams	Type hole
7....	36	B-34	0 - 5	4,653	4,188	33.4	1.68	14.5	37.5	Bucket.
	36	B-35	5 - 10	4,468	3,798	23.8	1.68	20.0	20.4	Do.
	36	B-36	10 - 15	4,564	3,971	29.6	1.68	33.0	36.0	Do.
	36	B-37	15 - 20	4,806	4,085	36.4	1.68	97.5	109.1	Do.
	36	B-38	20 - 25	5,053	4,295	36.1	1.68	39.5	41.1	Do.
	36	B-39	25 - 30	7,408	6,519	37.8	1.68	27.8	29.7	Do.
	36	B-40	30 - 35	5,957	5,242	38.7	1.68	60.0	62.2	Do.
	36	B-41	35 - 40	6,423	5,781	40.0	1.68	88.5	91.2	Do.
	36	B-42	40 - 45	6,593	5,802	32.8	1.68	356.0	363.0	Do.
	36	B-43	45 - 50	5,999	5,399	38.1	1.68	356.5	365.3	Do.
	36	B-44	50 - 55	6,388	5,621	33.3	1.68	150.0	153.9	Do.
	36	B-45	55 - 56	764	649	25.1	.34	15.6	16.3	Do.
	5	C-6	56 - 92	721	671	15.8	.18	22.0	23.4	Rotary.
	5	C-7	92 -113.7	799	743	4.0	.11	4.6	4.8	Do.
8....	5	C-8	0 - 81.8	2,647	2,462	37.8	.41	53.3	55.5	Do.
	5	C-9	81.8-101.8	605	563	31.9	.10	134.0	134.0	Do.
9....	36	B-74	0 - 5	1,958	1,821	33.2	1.68	16.0	16.3	Bucket.
	36	B-75	5 - 10	5,918	5,196	34.8	1.68	392.0	395.8	Do.
	36	B-76	10 - 15	3,915	3,328	30.0	1.68	300.0	301.7	Do.
	36	B-77	15 - 20	4,141	3,661	29.1	1.68	76.0	76.9	Do.
	36	B-78	20 - 23	1,669	1,441	26.7	1.01	120.0	120.5	Do.
	30	B-93	23 - 25	1,358	1,188	14.7	.31	23.1	23.6	Do.
	30	B-94	25 - 30	4,187	3,588	15.3	.78	25.6	26.3	Do.
	30	B-95	30 - 35	3,844	3,267	26.3	.78	48.0	48.3	Do.
	30	B-96	35 - 40	5,795	4,932	26.0	.78	126.0	128.5	Do.
	30	B-97	40 - 45	3,181	2,704	13.5	.78	59.0	59.2	Do.
	30	B-98	45 - 50	4,896	4,162	29.1	.78	58.0	59.6	Do.
	30	B-99	50 - 55	4,833	4,108	31.9	.78	31.0	31.6	Do.
	30	B-100	55 - 60	3,939	3,348	34.3	.78	32.0	32.6	Do.
	30	B-101	60 - 65	4,634	3,939	34.7	.78	23.2	24.6	Do.
	30	B-102	65 - 70	4,431	3,766	11.4	.78	33.0	33.8	Do.
	30	B-103	70 - 75	3,234	2,749	26.3	.78	27.0	28.9	Do.
	30	B-104	75 - 80	5,064	4,305	25.7	.78	92.0	93.5	Do.
	30	B-105	80 - 85	4,002	3,406	25.1	.78	91.0	94.2	Do.
	30	B-106	85 - 90	4,472	3,850	43.4	.78	27.8	30.4	Do.
	30	B-107	90 - 95	3,814	3,242	36.1	.78	27.2	30.3	Do.
	30	B-108	95 -100	4,279	3,637	37.2	.78	69.0	72.5	Do.
	30	B-109	100 -105	3,479	2,957	38.5	.78	43.0	44.3	Do.
	30	B-110	105 -109	2,872	2,599	44.3	.62	49.0	51.4	Do.
	5	C-15	111.2-137	1,059	985	32.7	.13	68.0	70.2	Rotary.
	5	C-16	137 -160	676	629	21.9	.12	8.2	9.3	Do.
10...	36	B-46	0 - 5	4,382	3,970	29.3	1.68	63.0	66.2	Bucket.
	36	B-47	5 - 10	5,471	5,060	32.8	1.68	79.0	80.7	Do.
	36	B-48	10 - 15	4,849	4,466	9.6	1.68	6.3	6.7	Do.

See footnotes at end of table.

TABLE A-1. - Drill hole sample data--Continued

Hole	Diameter, inches	Sample	Sample interval, feet	Net weight, pounds	Dry weight, pounds	Plus 1/8-inch fraction, percent	Approximate volume, cubic yards	Gold recovery, milligrams	Total gold,[1] milligrams	Type hole
10...	36	B-49	15 - 20	4,809	4,386	23.9	1.68	33.0	34.2	Bucket.
	36	B-50	20 - 26	4,953	4,606	28.8	2.02	142.0	143.0	Do.
11...	36	B-51	0 - 5	4,296	3,690	6.2	1.68	42.0	43.3	Do.
	36	B-52	5 - 10	5,102	4,337	1.5	1.68	17.0	17.5	Do.
	36	B-53	10 - 15	4,326	3,677	.1	1.68	26.9	26.9	Do.
	36	B-54	15 - 20	3,734	3,174	-	1.68	Nil	Nil	Do.
	36	B-55	20 - 25	3,924	3,335	10.8	1.68	49.0	50.6	Do.
	36	B-56	25 - 30	4,698	4,181	40.5	1.68	173.0	186.1	Do.
	36	B-57	30 - 35	4,794	4,406	22.9	1.68	68.0	69.4	Do.
	36	B-58	35 - 40	4,475	4,171	30.6	1.68	63.0	65.1	Do.
	36	B-59	40 - 45	4,319	3,969	23.6	1.68	30.0	30.0	Do.
	36	B-60	45 - 50	5,131	4,715	29.4	1.68	42.0	42.9	Do.
	36	B-61	50 - 55	4,413	4,033	22.1	1.68	56.0	57.7	Do.
	36	B-62	55 - 60	4,880	4,387	19.9	1.68	56.0	57.4	Do.
	30	B-63	60 - 65	2,839	2,467	13.2	.78	17.8	18.3	Do.
	30	B-64	65 - 70	3,465	2,994	6.5	.78	21.0	21.6	Do.
	30	B-65	70 - 75	4,228	3,640	2.0	.78	11.6	12.1	Do.
	30	B-66	75 - 77	2,131	1,811	4.9	.31	12.7	13.0	Do.
12...	30	B-130	0 - 5	2,589	2,382	30.4	.78	31.0	32.0	Do.
	30	B-131	5 - 10	3,630	3,220	28.8	.78	32.0	32.4	Do.
	30	B-132	10 - 15	2,794	2,442	34.1	.78	11.7	12.0	Do.
	30	B-133	15 - 20	2,799	2,390	34.2	.78	12.5	13.0	Do.
	30	B-134	20 - 25	4,444	3,782	32.0	.78	35.0	36.0	Do.
	30	B-135	25 - 30	3,351	2,848	40.6	.78	63.0	63.6	Do.
	30	B-136	30 - 35	3,454	2,946	35.6	.78	10.0	10.5	Do.
	30	B-137	35 - 40	3,742	3,181	30.9	.78	56.0	59.6	Do.
	30	B-138	40 - 45	3,887	3,304	29.8	.78	50.0	53.4	Do.
	30	B-139	45 - 50	3,862	3,283	36.9	.78	51.0	53.6	Do.
	30	B-140	50 - 55	3,476	2,955	29.5	.78	32.0	35.5	Do.
	30	B-141	55 - 60	4,415	3,753	37.6	.78	32.0	33.5	Do.
	30	B-142	60 - 65	4,319	3,671	37.6	.78	39.0	41.1	Do.
	30	B-143	65 - 70	3,441	2,925	36.3	.78	56.0	58.0	Do.
	30	B-144	70 - 75	4,190	3,562	33.0	.78	24.0	25.7	Do.
	30	B-145	75 - 80	3,746	3,184	15.3	.78	36.0	38.8	Do.
	30	B-146	80 - 85	4,631	3,969	37.2	.78	14.7	17.7	Do.
	30	B-147	85 - 90	3,625	3,128	42.0	.78	33.0	35.8	Do.
	30	B-148	90 - 94	4,018	3,468	46.2	.62	24.0	27.1	Do.
	5	C-20	94 -152	908	844	24.3	.29	15.2	([2])	Rotary.
	5	C-21	152 -208	1,086	1,010	20.0	.28	34.0	36.2	Do.
12A..	30	B-67	0 - 5	3,157	2,829	28.7	.78	97.0	98.4	Bucket.
	30	B-68	5 - 10	2,930	2,555	26.3	.78	160.0	161.6	Do.
	30	B-69	10 - 15	3,216	2,882	35.1	.78	105.0	107.9	Do.
	30	B-70	15 - 20	3,984	3,629	37.9	.78	30.5	31.8	Do.
	30	B-71	20 - 25	3,122	2,844	39.1	.78	53.0	53.8	Do.

See footnotes at end of table.

TABLE A-1. - Drill hole sample data--Continued

Hole	Diameter, inches	Sample	Sample interval, feet	Net weight, pounds	Dry weight, pounds	Plus 1/8-inch fraction, percent	Approximate volume, cubic yards	Gold recovery, milligrams	Total gold,[1] milligrams	Type hole
12A..	30	B-72	25 - 30	3,485	3,164	49.9	0.78	42.0	42.6	Bucket.
	30	B-73	30 - 35	4,504	3,968	48.1	.78	100.0	166.3	Do.
12B..	30	B-79	0 - 5	1,656	1,426	26.4	.78	42.0	43.0	Do.
	30	B-80	5 - 10	2,831	2,409	23.1	.78	105.0	111.2	Do.
	30	B-81	10 - 15	2,727	2,364	15.5	.78	66.0	67.4	Do.
	30	B-82	15 - 20	3,022	2,696	25.9	.78	90.0	91.8	Do.
	30	B-83	20 - 25	3,861	3,544	34.3	.78	(2)	(2)	Do.
	30	B-84	25 - 30	4,072	3,685	53.5	.78	417.0	418.0	Do.
	30	B-85	30 - 35	3,894	3,547	51.5	.78	59.0	62.6	Do.
	30	B-86	35 - 36.5	1,614	1,482	51.4	.23	65.0	68.8	Do.
13...	36	B-111	0 - 5	3,312	3,070	25.9	1.68	31.3	39.2	Do.
	36	B-112	5 - 10	4,283	3,799	15.4	1.68	57.0	128.7	Do.
	36	B-113	10 - 15	5,709	4,875	28.0	1.41	51.0	56.5	Do.
	30	B-114	15 - 20	4,463	3,843	24.5	.78	24.0	30.1	Do.
	30	B-115	20 - 25	3,225	2,806	33.1	.78	12.3	13.4	Do.
	30	B-116	25 - 30	4,819	4,154	37.[illegible]	.78	28.0	29.6	Do.
	30	B-117	30 - 35	3,479	3,865	34.6	.78	21.0	21.9	Do.
	30	B-118	35 - 40	3,919	3,464	32.[illegible]	.78	28.0	29.1	Do.
	30	B-119	40 - 45	3,074	2,681	23.[illegible]	.78	48.0	49.6	Do.
	30	B-120	45 - 50	3,333	2,853	17.9	.78	16.6	17.0	Do.
	30	B-121	50 - 55	3,492	2,993	29.0	.78	42.0	44.2	Do.
	30	B-122	55 - 60	5,529	4,722	26.1	.78	88.0	90.2	Do.
	30	B-123	60 - 65	3,292	2,821	31.5	.78	32.0	33.6	Do.
	30	B-124	65 - 70	4,844	4,166	28.4	.78	63.0	63.6	Do.
	30	B-125	70 - 75	3,437	2,925	30.2	.78	75.0	77.3	Do.
	30	B-126	75 - 80	4,210	3,646	25.2	.78	54.0	57.3	Do.
	30	B-127	80 - 85	3,955	3,378	29.7	.78	95.0	97.4	Do.
	30	B-128	85 - 90	4,492	3,917	49.3	.78	16.5	17.3	Do.
	30	B-129	90 - 91	1,303	1,144	57.[illegible]	.16	9.4	10.2	Do.
	30	C-17	91 - 147	937	871	7.2	.[illegible]8	12.3	12.6	Rotary.
	30	C-18	147 - 167	654	608	.9	.1[illegible]	1.6	2.0	Do.
	30	C-19	167 - 222	222	1,120	5.8	.28	4.0	4.2	Do.
14...	30	B-149	0 - 5	2,401	2,187	27.4	.78	27.0	28.4	Bucket.
	30	B-150	0 - 10	2,629	2,487	23.1	.78	12.3	12.8	Do.
	30	B-151	10 - 13	3,057	2,877	19.9	.47	27.0	28.4	Do
	30	B-152	13 - 20	2,476	2,273	5.9	1.10	19.5	19.8	Do.
	30	B-153	20 - 25	2,892	2,643	10.0	.78	22.	22.7	Do.
	30	B-154	25 - 30	2,258	2,037	7.1	.78	29.0	29.3	Do.
	30	B-155	30 - 35	2,960	2,697	3.8	.78	15.2	15.5	Do.
	30	B-156	35 - 40	2,703	2,454	10.0	.78	5.7	6.1	Do.
	30	B-157	40 - 45	2,800	2,484	12.2	.78	23.0	23.5	Do.
	30	B-158	45 - 50	3,086	2,666	9.7	.78	17.1	17.5	Do.
	30	B-159	50 - 55	4,893	4,159	3.8	.78	15.8	16.1	Do.
	30	B-160	55 - 60	1,316	1,119	1.6	.16	7.0	7.2	Do.

See footnotes at end of table.

TABLE A-1. - Drill hole sample data--Continued

Hole	Diameter, inches	Sample	Sample interval, feet	Net weight, pounds	Dry weight, pounds	Plus 1/8-inch fraction, percent	Approximate volume, cubic yards	Gold recovery, milligrams	Total gold,[1] milligrams	Type hole
15...	30	B-87	0 - 5	3,339	3,002	43.5	0.78	95.0	101.9	Bucket.
	30	B-88	5 - 10	3,586	3,288	63.1	.78	50.0	50.9	Do.
	30	B-89	10 - 15	3,641	3,372	57.8	.78	52.0	54.0	Do.
	30	B-90	15 - 20	4,095	3,817	64.0	.78	85.0	86.1	Do.
	30	B-91	20 - 25	3,802	3,528	67.5	.78	182.0	183.9	Do.
	30	B-92	25 - 29	3,681	3,375	62.3	.62	230.0	232.6	Do.
	5	C-12	31.5-111.5	2,341	2,177	42.2	.40	180.0	185.0	Rotary.
	5	C-13	111.5-133.0	707	658	30.0	.11	152.0	154.9	Do.
VH-1..	5	C-14	0 -121	5,910	5,142	27.8	1.37	640.0	642.2	Ventilation.
A.....	7	HH-1	0 -120	4,695	4,287	29.9	1.19	121.58	([3])	Hydral.
	7	HH-2	120 -140	682	634	28.3	.20	60.34	([3])	Do.
B.....	7	HH-3	0 -120	([2])	([2])	([2])	([2])	([2])	([3])	Do.
	7	HH-4	120 -143	875	814	41.7	.23	156.96	([3])	Do.
C.....	7	HH-5	0 -120	2,173	1,984	34.8	1.19	36.92	([3])	Do.
	7	HH-6	120 -145	686	638	43.3	.25	47.68	([3])	Do.
D.....	7	HH-7	0 -190	8,509	7,556	11.9	1.88	27.45	([3])	Do.
	7	HH-8	190 -212	864	804	33.7	.22	11.00	([3])	Do.
E.....	7	HH-9	0 -260	6,516	5,786	11.1	5.25	10.39	([3])	Do.
	7	HH-10	260 -295	547	509	11.0	.71	41.09	([3])	Do.
BL-1..	5	1	0 - 57	([2])	([2])	([2])	([2])	([2])	([3])	Blast.
BL-2..	5	2	0 - 32	1,097	976	23.9	.16	79.90	([3])	Do.
BL-3..	5	3	0 - 32	1,049	1,020	27.4	.16	73.00	([3])	Do.
BL-4..	5	4	0 - 32	638	593	32.6	.16	38.38	([3])	Do.
BL-5..	5	5	0 - 32	709	659	38.5	.16	57.84	([3])	Do.
BL-6..	5	6	0 - 48.5	1,418	1,319	48.6	.16	126.90	([3])	Do.
BL-7..	5	7	0 - 35	1,273	1,184	37.2	.24	83.56	([3])	Do.
BL-8..	5	8	0 - 35	1,348	1,254	37.9	.18	64.6[illegible]	([3])	Do.
BL-9..	5	9	0 - 35	1,150	1,070	36.8	.18	76.75	([3])	Do.
BL-10.	5	10	0 - 31.8	994	924	39.0	.16	81.85	([3])	Do.
BL-11	5	11	0 - 31.8	1,080	1,004	43.9	.16	66.94	([3])	Do.
BL-12	5	12	0 - 29	1,105	1,028	40.6	.15	140.20	([3])	Do.

[1] Includes gold recovered by cyanidation of amalgam residue.
[2] No data.
[3] Cyanidation not conducted on samples from holes A through BL-12.

TABLE A-2. - Underground sample data

Location	Sample	Net weight, pounds	Dry weight, pounds	Plus 1/8-inch fraction, percent	Approximate volume, cubic yards	Gold recovery, milli-grams	Total gold,[1] milli-grams	Type sample
Adit rooms.....	AR-1	2,588	2,347	59.7	0.66	520.0	535.9	Blast round
	AR-2	641	571	53.8	.16	244.0	248.5	grab.
	AR-3	819	753	83.9	.21	310.0	311.6	Do.
	AR-4	545	494	56.9	.14	210.0	212.1	Do.
	AR-5	650	590	51.4	.17	290.0	290.9	Do.
	AR-6	912	827	54.6	.23	349.0	354.0	Do.
	AR-7	507	468	59.0	.13	150.0	154.5	Do.
	AR-8	775	704	59.4	.20	290.0	291.3	Do.
	AR-9	1,752	1,545	60.0	.45	660.0	663.1	Do.
	AR-10	1,729	1,568	61.1	.44	290.0	293.1	Do.
	AR-11	1,625	1,474	42.5	.42	230.0	231.0	Do.
	AR-12	611	554	54.3	.16	192.0	194.4	Do.
	AR-13	654	593	59.2	.17	140.0	154.3	Do.
	AR-14	642	582	60.6	.16	380.0	381.1	Do.
	AR-15	868	787	56.3	.22	175.0	176.8	Do.
	AR-16	948	860	57.5	.24	169.0	171.9	Do.
	AR-17	750	680	61.5	.19	260.0	276.3	Do.
	AR-18	7,587	6,881	60.1	1.95	1,987.77	([2])	Do.
	AR-19	3,776	3,425	73.5	.96	800.0	830.3	Do.
	AR-20	3,817	3,462	74.6	.98	652.0	652.8	Do.
	AR-21	3,839	3,482	73.4	.98	678.0	682.1	Do.
	AR-22	3,638	3,300	77.1	.93	630.0	654.4	Do.
	AR-23	3,389	3,074	73.5	.87	490.0	604.0	Do.
	AR-24	3,320	3,011	73.5	.85	532.0	545.3	Do.
	AR-25	3,325	3,016	85.9	.85	643.0	655.6	Do.
	AR-26	3,554	3,223	67.8	.91	322.0	326.3	Do.
	AR-27	3,826	3,432	72.2	.98	490.0	509.0	Do.
	AR-28	3,600	3,258	72.6	.92	1,635.0	1,640.8	Do.
	AR-29	3,358	2,965	73.3	.86	600.0	604.0	Do.
	AR-30	3,413	3,171	76.5	.87	340.0	344.8	Do.
	AR-31	3,554	3,202	73.0	.91	370.0	376.2	Do.
	AR-32	7,048	6,364	72.5	1.81	610.0	640.6	Do.
	AR-33	3,56[illegible]	3,219	70.4	.91	360.0	377.6	Do.
	AR-34	3,706	3,354	72.6	.95	298.0	304.2	Do.
	AR-35	3,739	3,496	78.6	.96	580.0	587.4	Do.
	AR-36	3,713	3,397	80.3	.95	600.0	615.3	Do.
	AR-37	3,784	3,421	77.0	.97	950.0	955.8	Do.
	AR-38	3,878	3,572	76.3	.99	805.0	817.2	Do.
Upper level....	1	42.5	39.5	89.4	.011	8.0	([2])	Channel.
	2	48.0	44.6	96.9	.012	5.0	([2])	Do.
	3	50.0	46.5	91.0	.013	11.0	([2])	Do.
	4	56.0	52.1	92.0	.014	23.0	([2])	Do.
	5	60.0	55.8	86.7	.015	11.0	([2])	Do.
	6	45.5	42.3	87.9	.012	17.0	([2])	Do.
	7	51.0	47.4	72.5	.013	24.0	([2])	Do.
	8	47.5	44.2	94.7	.012	44.0	([2])	Do.
	9	59.5	55.3	74.8	.015	67.0	([2])	Do.
Bedrock drift..	1	103	95.8	68.9	.026	64.0	([2])	Do.
	2	100	93	63.0	.026	69.0	([2])	Do.
	3	121	113	48.8	.031	227.0	([2])	Do.

[1] Includes gold recovered by cyanidation of amalgam residue.
[2] Samples not amalgamated.

TABLE A-3. - Surface pit sample data

Location	Sample	Net weight, pounds	Dry weight, pounds	Plus 1/8-inch fraction, percent	Approximate volume, cubic yards	Gold recovery, milligrams	Total gold,[1] milligrams	Type sample
Toe lower bench.	TB-1	4,510	4,194	57.1	1.16	846.09	-	Toe bulk.
	TB-2	2,930	2,930	74.2	.75	130.0	134.1	Do.
	TB-3	2,768	2,574	72.6	.71	280.0	291.4	Do.
	TB-4	2,742	2,550	77.8	.70	187.0	192.3	Do.
	TB-5	2,892	2,690	76.5	.74	260.0	266.4	Do.
	TB-6	2,979	2,770	75.4	.76	130.0	134.8	Do.
	TB-7	2,999	2,789	70.5	.77	214.0	217.5	Do.
Lower bench stratigraphic.	T-1	256	246	59.5	.065	16.2	17.2	Stratigraphic.
	T-2	426	409	51.9	.109	14.0	15.6	Do.
	T-3	396	380	59.3	.102	15.0	15.7	Do.
	T-4	118	113	11.9	.030	1.6	1.9	Do.
	T-5	122	117	36.1	.031	2.4	2.6	Do.
	T-6	122	117	42.6	.031	7.2	7.5	Do.
	T-7	120	115	51.7	.031	1.5	1.8	Do.
	T-8	125	120	52.8	.032	4.8	5.2	Do.
	T-9	116	111	52.6	.030	7.4	7.9	Do.
	T-10	127	122	63.0	.033	6.4	6.6	Do.
	T-11	127	122	75.6	.033	([2])	([2])	Do.
	T-12	123	118	72.4	.032	12.6	12.7	Do.
	T-13	121	116	63.6	.031	8.1	12.6	Do.
Pit 1..........	A-1	2,742	2,550	51.7	1.00	100.64	108.09	Measured pit.
	P-1-R	112	104	([3])	.03	3.80	4.10	Do.
	P-1-F	111	103	([3])	.03	5.30	5.90	Do.
	P-1-L	117	109	([3])	.03	3.90	4.20	Do.
Pit 2..........	A-2	2,662	2,476	44.2	1.00	121.48	125.62	Do.
	P-2-R	119	111	([3])	.03	1.50	1.60	Do.
	P-2-F	111	103	([3])	.03	4.00	4.30	Do.
	P-2-L	116	108	([3])	.03	5.30	5.30	Do.
Pit 3..........	A-3	3,289	3,059	44.7	1.00	90.00	109.00	Do.
	P-3-R	117	109	([3])	.03	1.60	1.80	Do.
	P-3-F	120	112	([3])	.03	2.60	4.00	Do.
	P-3-L	112	104	([3])	.03	.50	.50	Do.
Pit 4..........	A-4	3,025	2,813	52.6	1.00	37.50	39.00	Do
	P-4-R	107	100	([3])	.03	.70	.90	Do
	P-4-F	98	91	([3])	.03	2.80	2.80	Do.
	P-4-L	113	105	([3])	.03	3.20	3.70	Do.
Pit 5..........	A-5	2,986	2,777	58.1	1.00	15.50	16.20	Do.
	P-5-R	103	96	([3])	.03	Trace	Trace	Do.
	P-5-L	120	112	([3])	.03	1.90	2.10	Do.
	P-5-F	109	101	([3])	.03	3.30	3.30	Do.

[1] Includes gold recovered by cyanidation of amalgam residue.
[2] Sample lost.
[3] Oversize not collected in "P" series.

TABLE A-4. - Gold particle size distribution (color count)--drill holes and pits

Hole	Sample	Particle size, millimeters[1]					Gold weight, milligrams
		+3	+2	+1	+0.5	-0.5	
1	B-1	-	-	3	14	29	72.50
	B-2	-	-	1	9	41	85.64
	B-3	-	-	-	12	38	63.43
	B-4	-	-	-	10	27	47.87
	B-5	-	-	7	23	32	113.73
	B-6	-	-	9	19	51	121.19
	B-7	-	-	-	-	-	-
	C-10	-	-	-	10	3	57.00
	C-11	-	-	-	-	3	4.50
2	B-8	-	-	11	42	110	265.04
	B-9	-	-	8	37	150	265.10
	B-10	-	-	-	7	53	79.69
	B-11	-	-	6	21	39	136.15
	B-12	-	1	5	37	127	251.09
	B-13	-	-	-	-	-	43.71
3	B-14	-	-	-	23	41	184.79
	B-15	-	-	-	3	22	17.16
	B-16	-	-	-	5	49	62.35
	C-3	-	-	6	14	28	200.00
4	B-17	-	-	-	2	12	22.40
	B-18	-	-	2	35	110	222.15
	B-19	-	-	10	63	240	508.38
	B-20	-	-	-	27	55	121.37
	B-21	-	-	-	35	49	112.49
	B-22	-	-	-	26	59	128.74
	B-23	-	-	2	23	30	121.35
	B-24	-	-	7	37	56	153.40
	B-25	-	-	-	8	26	57.82
5	B-26	-	-	7	52	160	306.92
	B-27	-	-	-	10	23	60.50
	B-28	-	-	-	3	25	45.50
	B-29	-	-	-	2	30	43.50
	B-30	-	-	-	1	14	8.00
	C-1	-	-	12	39	28	264.00
	C-2	-	-	5	11	19	185.00
6	B-31	-	-	-	7	55	95.50
	B-32	-	-	5	5	52	230.00
	B-33	-	-	1	4	12	28.00
	C-4	-	-	3	17	30	177.50
	C-5	-	-	6	8	6	200.00

See footnotes at end of table.

TABLE A-4. - Gold particle size distribution (color count)--drill holes and pits--Continued

Hole	Sample	Particle size, millimeters: 3	2	1	0.5	-0.5	Gold weight, milligrams
7	B-34	-	-	-	-	-	14.5
	B-35	-	-	1	3	7	20.0
	B-36	-	-	1	3	1	33.0
	B-37	-	-	-	2	23	97.5
	B-38	-	-	2	3	10	39.5
	B-39	-	-	1	2	-	[illegible]
	B-40	-	-	3	8	30	65.5
	B-41	-	-	1	13	21	88.5
	B-42	-	-	3	-	54	356.0
	B-43	-	-	8	11	31	356.5
	B-44	-	-	4	10	56	150.0
	B-45	-	-	1	6	14	15.6
	C-6	-	-	-	5	16	22.6
	C-7	-	-	-	2	3	[illegible]
8	C-8	-	5	2	23	17	53.3
	C-9	-	-	4	-	20	134.0
9	B-74	-	-	1	1	6	16.0
	B-75	-	6	10	28	84	392.0
	B-76	-	2	3	50	36	300.0
	B-77	-	-	-	60	39	76.5
	B-78	-	-	2	23	59	126.0
	B-93	-	-	-	-	170	23.1
	B-94	-	1	2	-	52	25.6
	B-95	-	-	5	75	80	48.0
	B-96	-	-	-	2	200	126.0
	B-97	-	-	-	-	175	59.0
	B-98	-	-	-	10	210	58.0
	B-99	-	-	-	25	200	31.0
	B-100	-	-	-	-	135	32.0
	B-101	-	-	-	-	125	23.2
	B-102	-	-	-	-	110	33.0
	B-103	-	-	-	5	110	27.0
	B-104	-	-	20	100	200	97.0
	B-105	-	-	1	-	250	91.0
	B-106	-	-	-	10	120	27.8
	B-107	-	-	-	3	130	27.2
	B-108	-	-	-	-	240	69.0
	B-109	-	-	-	12	180	43.0
	B-110	-	-	-	-	125	49.0
	C-15	-	-	5	25	125	68.0
	C-16	-	-	-	-	30	8.2

See footnotes at end of table.

TABLE A-4. - Gold particle size distribution (color count)--drill holes and pits--Continued

Hole	Sample	Particle size, millimeters: +3	+2	+1	+0.5	-0.5	Gold weight, milligrams
10	B-46	1	-	-	6	-	63.0
	B-47	-	-	-	20	-	79.0
	B-48	-	-	-	-	24	6.3
	B-49	-	-	-	10	23	33.0
	B-50	-	-	-	60	72	142.0
11	B-51	-	-	5	15	21	42.0
	B-52	-	-	-	10	17	17.0
	B-53	-	-	2	5	13	26.9
	B-54	-	-	-	-	5	Nil
	B-55	-	-	20	-	-	49.0
	B-56	-	-	35	-	31	173.0
	B-57	-	-	5	-	19	68.0
	B-58	-	-	-	-	47	63.0
	B-59	-	-	3	-	23	30.0
	B-60	-	-	-	-	19	42.0
	B-61	-	-	-	8	30	56.0
	B-62	-	-	10	-	18	56.0
	B-63	-	-	-	-	3	17.8
	B-64	-	-	-	4	3	21.0
	B-65	-	-	1	-	3	11.6
	B-66	-	-	3	-	9	12.7
[illegible]	B-130	-	-	-	2	230	31.0
	B-131	-	-	7	10	120	32.0
	B-132	-	-	-	1	85	11.7
	B-133	-	-	-	15	80	12.5
	B-134	-	-	-	-	105	35.0
	B-135	-	-	-	-	130	63.0
	B-136	-	-	-	20	50	10.0
	B-137	-	-	2	5	110	56.0
	B-138	-	-	-	2	230	50.0
	B-139	-	-	-	5	135	51.0
	B-140	-	-	1	5	110	32.0
	B-141	-	-	-	10	250	32.0
	B-142	-	1	10	25	200	39.0
	B-143	-	-	-	5	104	56.0
	B-144	-	-	-	-	105	24.0
	B-145	-	-	-	-	220	36.0
	B-146	-	-	8	14	100	14.7
	B-147	-	-	1	-	160	33.0
	B-148	-	1	2	5	75	24.0
	C-20	-	-	2	25	-	15.2
	C-21	-	-	-	8	45	34.0

See footnotes at end of table.

TABLE A-4. - Gold particle size distribution (color count)--drill holes and pits--Continued

Hole	Sample	Particle size, millimeters[1]					Gold weight,[2] milligrams
		+3	+2	+1	+0.5	-0.5	
2A	B-67	-	-	10	30	-	97.0
	B-68	-	-	10	100	30	160.0
	B-69	-	-	-	-	100	105.0
	B-70	-	-	-	12	150	30.5
	B-71	-	-	20	70	45	53.0
	B-72	-	-	20	75	35	42.0
	B-73	-	-	8	75	130	100.0
12B	B-79	-	-	-	15	125	42.0
	B-80	-	-	10	40	225	105.0
	B-81	-	-	2	30	200	66.0
	B-82	-	-	3	40	180	90.0
	B-83	-	-	-	-	-	-
	B-84	-	-	25	100	550	417.0
	B-85	-	1	5	30	125	59.0
	B-86	-	-	6	10	150	65.0
13	B-111	-	-	-	10	150	31.3
	B-112	-	-	3	5	70	57.0
	B-113	-	-	3	25	150	51.0
	B-114	-	-	-	-	150	24.0
	B-115	-	-	-	10	50	12.3
	B-116	-	-	-	50	75	28.0
	B-117	-	-	2	75	50	21.0
	B-118	-	-	-	-	118	28.0
	B-119	-	-	-	1	185	48.0
	B-120	-	-	-	-	130	16.6
	B-121	-	-	-	5	145	42.0
	B-122	-	-	-	2	240	88.0
	B-123	-	-	-	5	90	32.0
	B-124	-	-	5	25	150	63.0
	B-125	-	-	-	10	200	75.0
	B-126	-	-	1	5	175	54.0
	B-127	-	-	-	10	150	95.0
	B-128	-	-	-	-	120	16.5
	B-129	-	-	-	-	80	9.4
	C-17	-	-	-	12	70	12.3
	C-18	-	-	-	-	8	1.6
	C-19	-	-	-	-	18	4.0
[illegible]	B-149	-	-	1	1	5	27.0
	B-150	-	-	-	-	170	12.3
	B-151	-	-	-	-	250	27.0
	B-152	-	-	-	-	200	19.5
	B-153	-	-	-	-	40	22.5

See footnotes at end of table.

TABLE A-4. - Gold particle size distribution (color count)--drill holes and pits--Continued

Hole	Sample	Particle size, millimeters[1] +3	+2	+1	+0.5	-0.5	Gold weight,[2] milligrams
14	B-154	-	-	-	-	205	29.0
	B-155	-	-	-	-	126	15.2
	B-156	-	-	-	-	103	5.7
	B-157	-	-	-	-	260	23.0
	B-158	-	-	-	-	300	17.1
	B-159	-	-	-	-	75	15.8
	B-160	-	-	-	-	31	7.0
15	B-87	-	-	1	101	205	95.0
	B-88	-	-	3	10	150	50.0
	B-89	-	-	3	50	175	52.0
	B-90	-	-	2	10	175	85.0
	B-91	-	-	5	52	225	182.0
	B-92	-	-	12	120	130	230.0
	C-12	-	-	25	75	50	180.0
	C-13	-	1	35	50	100	152.0
VH-1	C-14	-	-	22	61	700	640.0
A	HH-1	-	-	5	109	290	121.6
	HH-2	-	-	6	35	89	60.3
B	HH-3	-	-	-	200	175	([3])
	HH-4	-	-	9	106	271	157.0
C	HH-5	-	-	1	29	74	36.9
	HH-6	-	-	5	33	99	47.7
D	HH-7	-	-	-	4	333	27.5
	HH-8	-	-	-	1	311	11.0
E	HH-9	-	-	-	5	63	10.4
	HH-10	-	-	-	15	80	41.1
BL-1	1	-	-	-	-	-	-
BL-2	2	-	-	5	79	273	79.9
BL-3	3	-	-	4	57	335	73.0
BL-4	4	-	-	-	24	224	38.4
BL-5	5	-	-	3	40	256	57.8
BL-6	6	-	-	2	111	334	126.9
BL-7	7	-	-	4	58	172	83.6
BL-8	8	-	-	4	30	231	64.7
BL-9	9	-	-	8	38	296	76.7
BL-10	10	-	-	4	43	270	81.9
BL-11	11	-	-	-	64	156	66.9
BL-12	12	-	-	7	96	348	140.2

See footnotes at end of table.

TABLE A-4. - Gold particle size distribution (color count)--drill holes and pits--Continued

Hole	Sample	Particle size, millimeters[1] +3	+2	+1	+0.5	-0.5	Gold weight,[2] milligrams
Pit 1..................	P-1-L	-	-	1	4	3	3.9
	P-1-R	-	-	4	11	8	3.8
	P-1-F	-	-	2	8	11	5.3
Pit 2..................	P-2-L	-	-	2	6	2	5.3
	P-2-R	-	-	-	-	10	1.5
	P-2-F	-	-	3	-	6	4.0
Pit 3..................	P-3-L	-	-	-	-	9	.5
	P-3-R	-	-	-	1	8	1.6
	P-3-F	-	-	-	5	2	2.6
Pit 4..................	P-4-L	-	-	-	1	9	3.2
	P-4-R	-	-	-	-	9	.7
	P-4-F	-	-	1	2	6	2.8
Pit 5..................	P-5-L	-	-	-	-	4	3.3
	P-5-R	-	-	1	-	5	.6
	P-5-F	-	-	1	1	3	1.9
Stratigraphic channel..	T-1	-	3	2	25	124	16.2
	T-2	-	-	7	14	30	14.0
	T-3	-	4	10	16	72	15.0
	T-4	-	-	-	-	3	1.6
	T-5	-	-	-	2	3	2.4
	T-6	-	-	-	10	10	7.2
	T-7	-	-	-	1	10	1.5
	T-8	-	-	-	16	3	4.8
	T-9	-	1	9	25	-	7.4
	T-10	-	-	5	-	-	6.4
	T-11	([3])	([3])	([3])	([3])	([3])	([3])
	T-12	-	-	2	-	10	12.6
	T-13	-	2	2	15	-	8.1

[1] Does not include flour gold.
[2] Amount recovered from concentrate by amalgamation.
[3] Sample lost.

TABLE A-5. - Geologic logs--drill holes

Hole	Stratigraphic interval, feet	Distance, feet	Material	Color
1	0 - 7.0	7.0	Gravel	Brown.
	7.0- 9.0	2.0	Gravel, sand	Do.
	9.0- 27.0	18.0	Sand, clay, gravel, water at 19.0 feet.	Do.
	27.0- 33.0	6.0	Gravel, sand	Do.
	33.0- 34.7	1.7	Gravel	Do.
	34.7- 76.6	41.9	Cemented gravel	Blue.
	76.6- 81.6	5.0	Cemented gravel, clay, sand.	Do.
	81.6- 86.6	5.0	Cemented gravel, sand	Do.
	86.6- 91.6	5.0	Cemented gravel	Do.
	91.6-101.6	10.0	Clay, sand	Do.
	101.6-103.6	2.0	Cemented gravel	Do.
	103.6-128.6	25.0	Clay, sand	Do.
	128.6-139.3	10.7	Phyllite (bedrock)	Gray.
2	0 - 25.0	25.0	Gravel, sand, water at 18.0 feet.	Brown.
	25.0- 25.4	.4	Cemented gravel	Blue.
3	0 - 7.6	7.6	Gravel, sand, clay	Brown.
	7.6- 15.2	7.6	Gravel, water at 15.0 feet.	Do.
	15.2- 76.2	61.0	Cemented gravel, sand	Blue.
	76.2-101.2	25.0	Gravel, sand	Do.
	101.2-107.2	6.0	Sand, gravel, hole caved at 107 feet.	Do.
4	0 - 15.0	15.0	Gravel	Brown.
	15.0- 41.4	26.4	Gravel, sand, water at 17.5 feet, cemented gravel at 41.4 feet.	Do.
5	0 - 20.0	20.0	Gravel, sand, clay	Do.
	20.0- 20.3	.3	Gravel	Do.
	20.3-132.6	112.3	Cemented gravel	Blue.
	132.6-138.2	5.6	Diorite (bedrock)	Gray.
6	0 - 5.0	5.0	Gravel, sand, clay	Brown.
	5.0- 10.0	5.0	Sand, clay, gravel	Do.
	10.0- 12.0	2.0	Cemented gravel	Blue.
	12.0- 17.0	5.0	Cemented gravel, sand	Do.
	17.0- 22.0	5.0	Cemented sand, gravel	Do.
	22.0- 27.0	5.0	Cemented sand, clay	Do.

TABLE A-5. - Geologic logs--drill holes--Continued

Hole	Stratigraphic interval, feet	Distance, feet	Material	Color
6	27.0- 32.0	5.0	Cemented sand, gravel, clay.	Blue.
	32.0- 82.0	50.0	Cemented sand, gravel.	Do.
	82.0-107.0	25.0	Cemented gravel, sand	Do.
	107.0-122.0	15.0	Cemented sand, gravel.	Do.
	122.0-137.0	15.0	Cemented gravel, sand	Do.
	137.0-139.0	2.0	Cemented sand, gravel, clay.	Do.
	139.0-150.5	11.5	Phyllite, quartzite (bedrock).	Gray.
7	0 - 5.0	5.0	Clay, sand, gravel...	Brown.
	5.0- 10.0	5.0	Clay, sand..........	Do.
	10.0- 13.0	3.0	Sand................	Do.
	13.0- 15.0	2.0	Clay, sand..........	Do.
	15.0- 35.0	20.0	Clay, sand, gravel, water at 20.0 feet.	Do.
	35.0- 40.0	5.0	Clay, gravel, sand...	Do.
	40.0- 44.0	4.0	Sand................	Do.
	44.0- 50.0	6.0	Clay, sand, gravel...	Do.
	50.0- 55.0	5.0	Sand, gravel, clay...	Do.
	55.0- 56.0	1.0	Gravel, sand, clay...	Do.
	56.0- 72.0	16.0	Sand................	Blue.
	72.0-111.0	39.0	Clay................	Do.
	111.0-133.0	22.0	Quartzite (bedrock)..	Gray.
8	0 - 17.9	17.9	Sand, gravel, clay, water at 3.0 feet.	Brown.
	17.9- 41.8	23.9	Cemented gravel......	Blue.
	41.8- 46.8	5.0	Cemented gravel, sand	Do.
	46.8- 61.8	15.0	Cemented gravel, sand, clay.	Do.
	61.8- 76.8	15.0	Cemented gravel, sand	Do.
	76.8- 81.8	5.0	Cemented gravel......	Do.
	81.8- 94.8	13.0	Cemented gravel, sand	Do.
	94.8- 99.8	5.0	Clay................	Green.
	99.8-101.8	2.0	Phyllite (bedrock)	Gray.
9	0 - 5.0	5.0	Gravel, sand, water at 5.0 feet.	Brown.
	5.0- 10.0	5.0	Sand, gravel, clay...	Do.
	10.0- 11.0	1.0	Sand, gravel........	Do.
	11.0- 25.0	14.0	Clay, sand, gravel...	Gray.
	25.0- 30.0	5.0	Gravel..............	Yellow.
	30.0- 33.7	3.7	Sand, gravel, clay...	Pink.

TABLE A-5. - Geologic logs--drill holes--Continued

Hole	Stratigraphic interval, feet	Distance, feet	Material	Color
9	33.7- 35.0	1.3	Gravel	Pink.
	35.0- 37.0	2.0	Sand, gravel, clay	Do.
	37.0- 40.0	3.0	Gravel	Gray.
	40.0- 45.0	5.0	Sand	Tan.
	45.0- 66.5	21.5	Gravel, clay, sand	Tan-gray.
	66.5- 73.0	6.5	Sand, clay	Gray.
	73.0-109.0	36.0	Clay, gravel, sand	Gray-blue.
	109.0-117.0	8.0	Cemented sand, gravel	Blue.
	117.0-137.0	20.0	Cemented gravel	Do.
	137.0-142.0	5.0	Cemented clay, sand, gravel.	Gray.
	142.0-143.0	1.0	Clay, sand	Do.
	143.0-146.0	3.0	Cemented gravel	Do.
	146.0-147.0	1.0	Clay, sand	Do.
	147.0-151.0	4.0	Cemented gravel	Do.
	151.0-152.0	1.0	Clay	Do.
	152.0-155.0	3.0	Cemented gravel	Do.
	155.0-165.0	10.0	Quartzite (bedrock)	Do.
10	0 - 5.0	5.0	Clay, sand	Red.
	5.0- 17.0	12.0	....do	Brown.
	17.0- 19.0	2.0	....do	White.
	19.0- 20.0	1.0	Clay, sand, gravel	Do.
	20.0- 26.0	6.0	Sand, clay, gravel (caved).	Do.
11	0 - 10.0	10.0	Clay, sand	Red.
	10.0- 22.0	12.0	Sand, clay	Tan.
	22.0- 28.0	6.0	Clay, sand, gravel	Brown.
	28.0- 55.0	27.0	Sand, clay, gravel	Tan.
	55.0- 56.0	1.0	Sand	Gray.
	56.0- 60.0	4.0	Sand, clay, gravel	Do.
	60.0- 65.0	5.0	Clay, sand	Do.
	65.0- 75.0	10.0	Sand, clay, gravel	Do.
	75.0- 77.0	2.0	Sand, gravel, water at 75.0 feet.	Do.
12	0 - 7.0	7.0	Gravel, sand	White.
	7.0- 24.0	17.0	Gravel, sand, clay	Buff.
	24.0- 50.0	26.0	Clay, sand, gravel, water at 20.0 feet.	Brown.
	50.0- 51.0	1.0	Gravel	Buff.
	51.0- 52.0	1.0	Clay	Do.
	52.0- 55.0	3.0	Sand, gravel	Do.
	55.0- 59.0	4.0	Gravel	Brown.
	59.0- 60.0	1.0	Sand	Do.

TABLE A-[illegible]. - Geologic logs--drill holes--Continued

Hole	Stratigraphic interval, feet	Distance, feet	Material	Color
2	60.0- 65.0	5.0	Gravel, clay	Buff.
	65.0- 72.0	7.0	Gravel, clay, sand	Brown.
	72.0- 74.0	2.0	Sand	Buff.
	74.0- 75.0	1.0	Gravel	Do.
	75.0- 80.0	5.0	Sand	Brown.
	80.0- 81.7	1.7	Gravel	Buff.
	81.7- 83.0	1.3	Clay	Do.
	83.0- 94.0	11.0	Gravel, clay, sand	Brown.
	94.0-107.0	13.0	Cemented gravel, clay, sand.	Blue.
	107.0-122.0	15.0	Gravel, clay, sand	Do.
	122.0-152.0	30.0	Cemented gravel, clay, sand.	Do.
	152.0-167.0	15.0	Cemented gravel	Do.
	167.0-182.0	15.0	Cemented gravel, clay.	Do.
	182.0-197.0	15.0	Clay, gravel	Gray.
	197.0-199.0	2.0	Gravel, clay	Do.
	199.0-210.0	11.0	Quartzite (bedrock)	Do.
2A	0 - 10.0	10.0	Sand, clay, gravel	Brown.
	10.0- 20.3	10.3	Clay, gravel, sand	Do.
	20.3- 28.8	8.5	Gravel, sand, clay, water at 28.0 feet.	Do.
	28.8- 34.4	5.6	Sand, gravel, clay	Do.
	34.4- 34.6	.2	Cemented gravel	Blue.
2B	0 - 9.0	9.0	Sand, clay, gravel	Brown.
	9.0- 14.0	5.0	Sand	Do.
	14.0- 16.5	2.5	Sand, clay, gravel	Do.
	16.5- 19.0	2.5	Sand	Do.
	19.0- 26.5	7.5	Clay, sand, gravel	Do.
	26.5- 31.6	5.1	Clay, gravel, sand	Do.
	31.6- 33.5	1.9	Sand, gravel, water at 32.0 feet.	Do.
	33.5- 36.5	3.0	Cemented gravel	Blue.
[illegible]	0 - 7.0	7.0	Sand, gravel	Gray.
	7.0- 11.5	4.5	Clay, gravel	Do.
	11.5- 15.0	3.5	Gravel, sand	Brown.
	15.0- 17.0	2.0	Gravel, sand, clay	Do.
	17.0- 18.0	1.0	Sand	Do.
	18.0- 33.9	15.9	Sand, gravel, clay	Do.
	33.9- 35.0	1.1	Sand	Do.
	35.0- 47.0	12.0	Gravel, sand, clay	Gray.
	47.0- 48.6	1.6	Clay	White

TABLE A-5. - Geologic logs--drill holes--Continued

Hole	Stratigraphic interval, feet	Distance, feet	Material	Color
13	48.6- 51.3	2.7	Clay, gravel	Gray.
	51.3- 69.0	17.7	Sand, gravel, clay	Tan.
	69.0- 86.5	17.5	Gravel, clay	Brown.
	86.5- 89.0	2.5	.....do	Blue.
	89.0-102.0	13.0	Cemented gravel	Do.
	102.0-111.0	9.0	Clay	Gray.
	111.0-113.0	2.0	Cemented gravel	Blue.
	113.0-122.0	9.0	Compacted clay, gravel.	Do.
	122.0-127.0	5.0	.....do	Brown.
	127.0-162.0	35.0	Clay	Blue.
	162.0-167.0	5.0	Shaley clay	Green.
	167.0-198.0	31.0	Clay, shale	Do.
	198.0-222.0	24.0	Quartzite (bedrock)	Blue.
14	0 - 10.0	10.0	Clay, gravel	Red.
	10.0- 25.0	15.0	Sand, gravel	Brown.
	25.0- 30.0	5.0	Clay, gravel	Do.
	30.0- 39.0	9.0	Sand	Do.
	39.0- 40.0	1.0	Gravel, sand	Do.
	40.0- 42.0	2.0	Sand, clay	Do.
	42.0- 47.0	5.0	Gravel, sand, clay	Do.
	47.0- 48.0	1.0	Gravel	Do.
	48.0- 56.0	8.0	Clay, water at 49.0 feet.	White.
15	0 - 22.3	22.3	Sand, clay, gravel	Brown.
	22.3- 59.5	37.2	Cemented gravel, water at 24.0 feet.	Blue.
	59.5- 61.5	2.0	Clay	Do.
	61.5- 66.5	5.0	Cemented gravel, clay	Do.
	66.5-138.0	71.5	Cemented gravel	Do.
	138.0-146.5	8.5	Talc (bedrock)	Gray.
A	0 - 31.0	31.0	Sand, clay, gravel, water at 18.0 feet.	Brown.
	31.0- 39.0	8.0	Gravel, sand, clay	Do.
	39.0- 40.0	1.0	Cemented gravel	Blue.
	40.0- 53.0	13.0	Gravel	Do.
	53.0- 54.0	1.0	Clay	Gray.
	54.0- 83.0	29.0	Loosely cemented gravel.	Blue.
	83.0-100.0	17.0	Clay	Green.
	100.0-139.0	39.0	Compacted gravel, clay.	Gray.
	139.0-146.0	7.0	Phyllite (bedrock)	Do.

TABLE A-5. - Geologic logs--drill holes--Continued

Hole	Stratigraphic interval, feet	Distance, feet	Material	Color
B....................	0 - 18.0	18.0	Gravel, sand, water at 10.0 feet.	Brown.
	18.0- 23.5	5.5	Gravel, clay, sand...	Do.
	23.5- 25.0	1.5	Clay.................	Do.
	25.0- 40.0	15.0	Gravel...............	Do.
	40.0-140.0	100.0	Cemented gravel,	Blue.
	140.0-143.0	3.0	Andesite (bedrock)	Gray.
C....................	0 - 5.0	5.0	Gravel, sand.........	Brown.
	5.0- 20.0	15.0	Gravel, sand, clay, water at 12.0 feet.	Do.
	20.0- 25.0	5.0	Cemented gravel, sand, clay.	Do.
	25.0- 30.0	5.0	Sand, clay, gravel...	Do.
	30.0- 41.0	11.0	Clay, gravel, sand...	Do.
	41.0- 47.0	6.0	Gravel, sand, clay...	Blue.
	47.0- 50.0	3.0	Clay, sand...........	Do.
	50.0- 95.0	45.0	Cemented gravel, sand, clay.	Do.
	95.0-141.5	46.5	Cemented gravel......	Do.
	141.5-145.0	3.5	Phyllite (bedrock)...	Gray.
D....................	0 - 2.0	2.0	Clay.................	Brown.
	2.0- 10.0	8.0	Clay, sand...........	Gray.
	10.0- 11.5	1.5	Gravel, clay, sand...	Do.
	11.5- 33.5	22.0	Clay, sand, water at 34.0 feet.	Do.
	33.5- 40.0	6.5	Cemented clay, sand, gravel.	Do.
	40.0- 45.0	5.0	Clay, sand...........	Do.
	45.0- 50.0	5.0	Cemented gravel......	Do.
	50.0- 55.0	5.0	Clay, sand...........	Do.
	55.0- 64.0	9.0	Cemented clay, sand, gravel.	White.
	64.0- 72.0	8.0	Clay, sand...........	Brown.
	72.0- 75.0	3.0	Gravel...............	Do.
	75.0-125.0	50.0	Gravel, clay, sand...	Do.
	125.0-128.0	3.0	Sand.................	Do.
	128.0-130.0	2.0	Gravel...............	Do.
	130.0-150.0	20.0	Sand.................	Do.
	150.0-158.0	8.0	Gravel, sand, clay...	Do.
	158.0-180.0	22.0	Cemented gravel, sand, clay.	Blue.
	180.0-207.0	27.0	Cemented sand, gravel, clay.	Do.
	207.0-212.0	5.0	Phyllite (bedrock)...	Gray.

TABLE A-5. - Geologic logs--drill holes--Continued

Hole	Stratigraphic interval, feet	Distance, feet	Material	Color
E..........................	0 - 46.0	46.0	Clay, gravel, sand...	Red-gray.
	46.0. 48.0	2.0	Clay.................	Gray.
	48.0-102.0	54.0	Clay, sand, gravel...	Do.
	102.0-122.0	20.0	Clay, gravel.........	Do.
	122.0-140.0	18.0	Gravel, clay.........	Do.
	140.0-153.0	13.0	Clay, gravel.........	Do.
	153.0-180.0	27.0	.....do..............	Yellow.
	180.0-218.0	38.0	Clay, gravel, sand...	Gray-tan.
	218.0-222.0	4.0	Cemented gravel......	Blue.
	222.0-227.0	5.0	Clay.................	Do.
	227.0-273.0	46.0	Cemented gravel, clay	Do.
	273.0-318.0	45.0	Cemented gravel......	Do.
	318.0-332.0	14.0	Granite (bedrock)....	Gray.
[illegible]H-1..................	0 - 32.0	32.0	Gravel, clay.........	Brown.
	32.0- 47.0	15.0	Cemented gravel, sand	Blue.
	47.0- 92.0	45.0	Cemented gravel......	Do.
	92.0-104.0	12.0	Cemented clay, gravel	Do.
	104.0-112.0	8.0	Cemented gravel......	Do.
	112.0-114.5	2.5	Sand.................	Do.
	114.5-121.0	6.5	Cemented gravel......	Do.
[illegible]-1....................	0 - 56.0	56.0	.....do..............	Do.
	56.0- 57.0	1.0	Phyllite (bedrock)...	Gray.
[illegible]-2....................	0 - 16.0	16.0	Cemented clay, gravel	Blue.
	16.0- 32.0	[illegible]	Cemented gravel......	Do.
[illegible]-3....................	0 - 28.0	[illegible]	.....do..............	Do.
	28.0- 32.0	4.0	Cemented gravel, clay	Do.
[illegible]-4....................	0 - 16.0	16.0	Cemented gravel......	Do.
	16.0- 23.0	7.0	Cemented clay, gravel	Do.
	23.0- 27.0	4.0	Cemented gravel......	Do.
	27.0- 32.0	5.0	Clay, gravel.........	Do.
[illegible]-5....................	0 - 28.0	28.0	Cemented gravel, clay	Do.
	28.0- 28.8	.8	Clay.................	Do.
	28.8- 32.0	3.2	Cemented gravel......	Do.
[illegible]-6....................	0 - 29.0	29.0	Clay, gravel.........	Do.
	29.0- 45.0	16.0	Cemented gravel......	Do.
	45.0- 47.0	2.0	Cemented gravel, clay	Do.
	47.0- 48.5	1.5	Phyllite (bedrock)...	Gray.

TABLE A-5. - Geologic logs--drill holes--Continued

Hole	Stratigraphic interval, feet	Distance, feet	Material	Color
[illegible]-7	0 - 13.0	13.0	Cemented gravel......	Blue.
	13.0- 16.0	3.0	Cemented gravel, clay	Do.
	16.0- 35.0	19.0	Cemented gravel......	Do.
[illegible]-8	0 - 11.0	11.0	.....do..............	Do.
	11.0- 15.0	4.0	Cemented gravel, clay	Do.
	15.0- 25.0	10.0	Cemented gravel......	Do.
	25.0- 27.0	2.0	Clay................	Do.
	27.0- 31.0	4.0	Cemented gravel......	Do.
	31.0- 35.0	4.0	Cemented gravel, sand, clay.	Do.
[illegible]-9	0 - 32.0	32.0	Cemented gravel......	Do.
	32.0- 35.0	3.0	Cemented gravel, clay	Do.
[illegible]-10	0 - 31.8	31.8	Compacted sand, clay, gravel.	Do.
[illegible]-11	0 - 27.0	27.0	.....do..............	Do.
	27.0- 31.8	4.8	Cemented gravel......	Do.
[illegible]-12	0 - 29.0	29.0	Compacted sand, clay gravel.	Do.
Stratigraphic channel:				
T-1	0 - 2.0	2.0	Gravel, clay, sandy, lens.	Red-tan.
T-2	2.0- 5.0	3.0	Coarse gravel, sand..	Brown.
T-3	5.0- 8.0	3.0	Gravel, sand, clay...	Do.
T-4	8.0- 9.0	1.0	Coarse gravel, sand..	Red.
T-5	9.0- 10.0	1.0	Sand, gravel........	Brown.
T-6	10.0- 11.0	1.0	Coarse gravel, sand, cobbles.	Do.
T-7	11.0- 12.0	1.0	.....do.............	Do.
T-8	12.0- 13.0	1.0	.....do.............	Do.
T-9	13.0- 14.0	1.0	.....do.............	Do.
T-10	14.0- 15.0	1.0	.....do.............	Do.
T-11	15.0- 16.0	1.0	.....do.............	Do.
T-12	16.0- 17.0	1.0	.....do.............	Do.
T-13	17.0- 18.0	1.0	Cemented gravel......	Do.

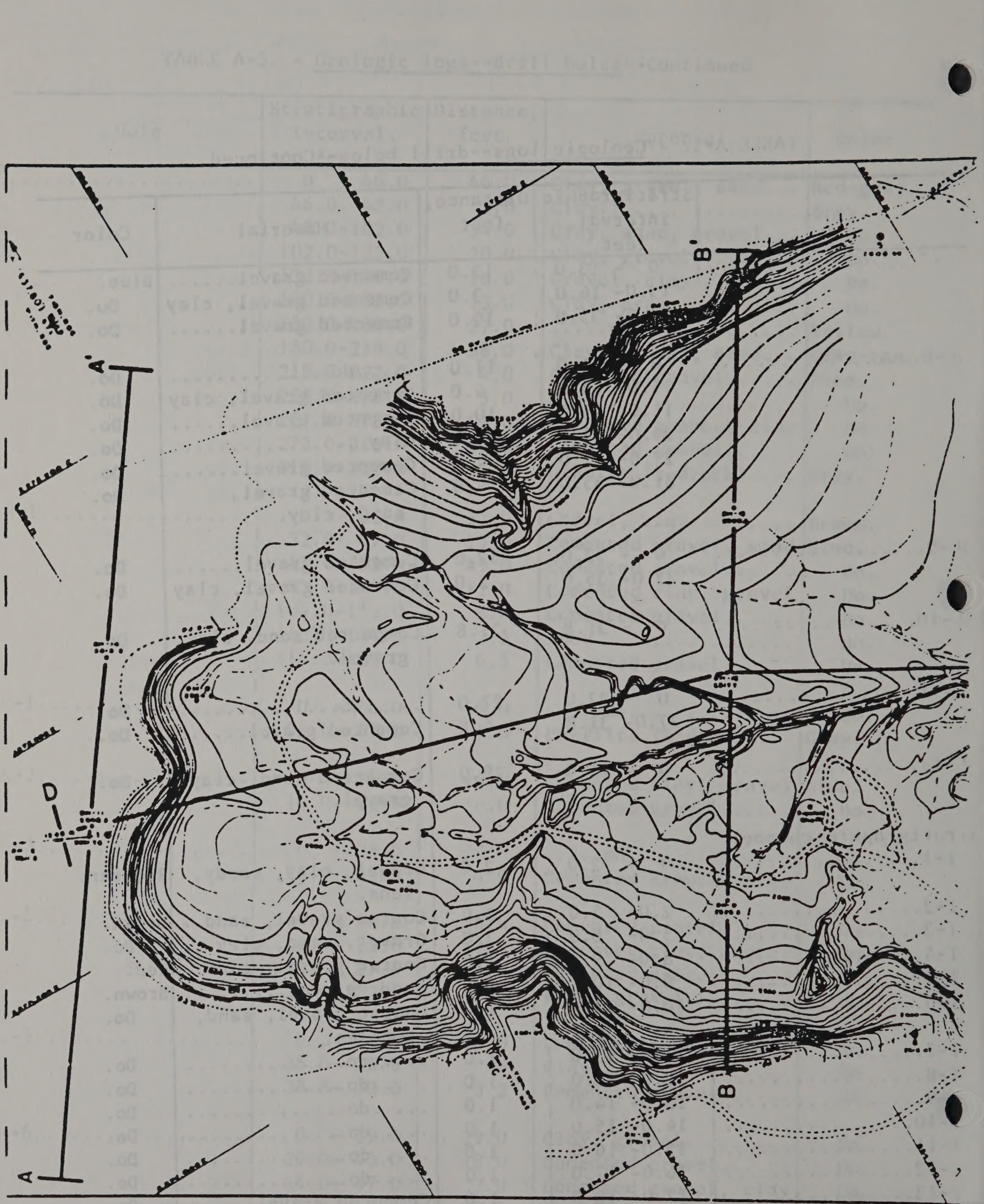


FIGURE 3.

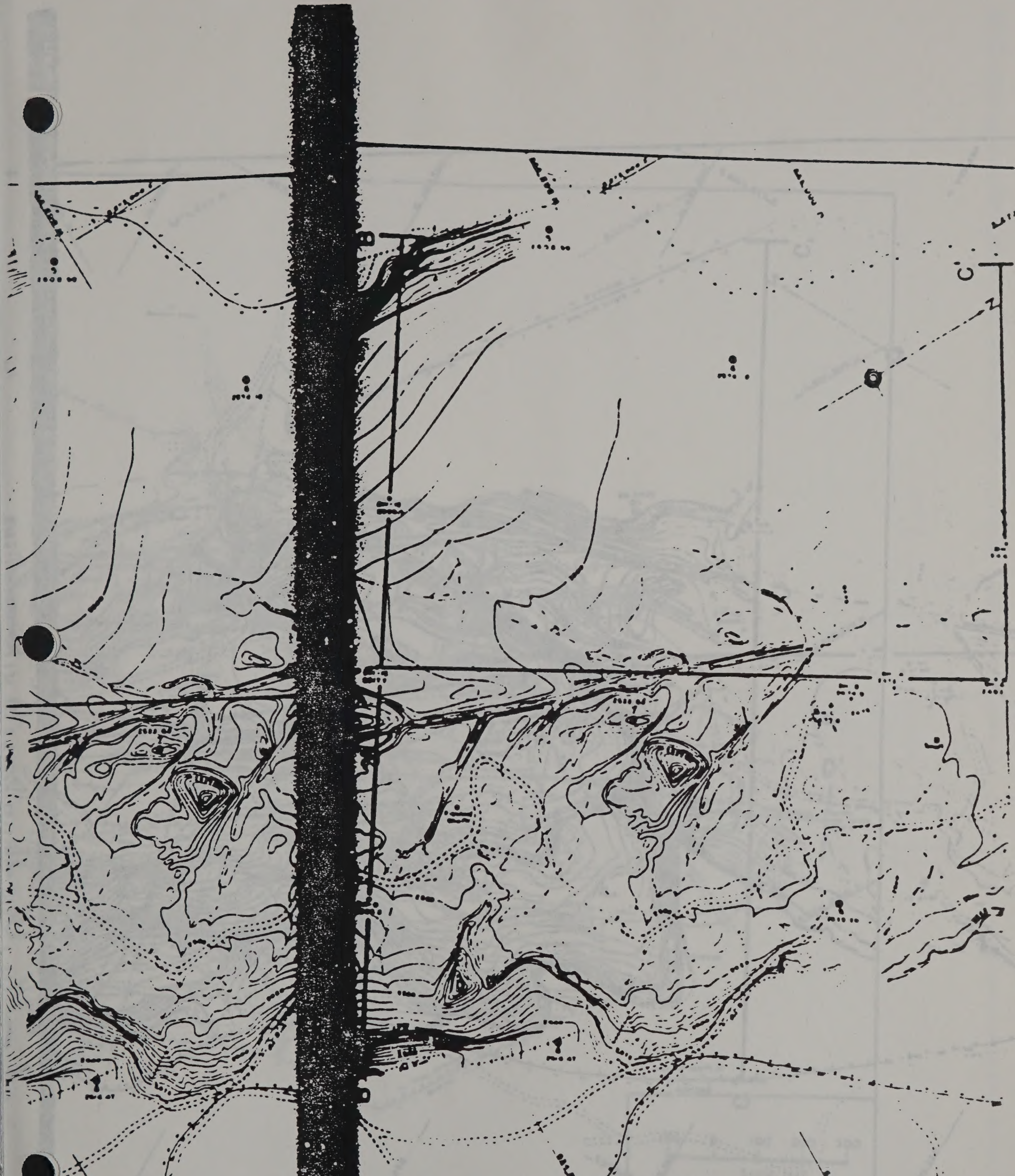
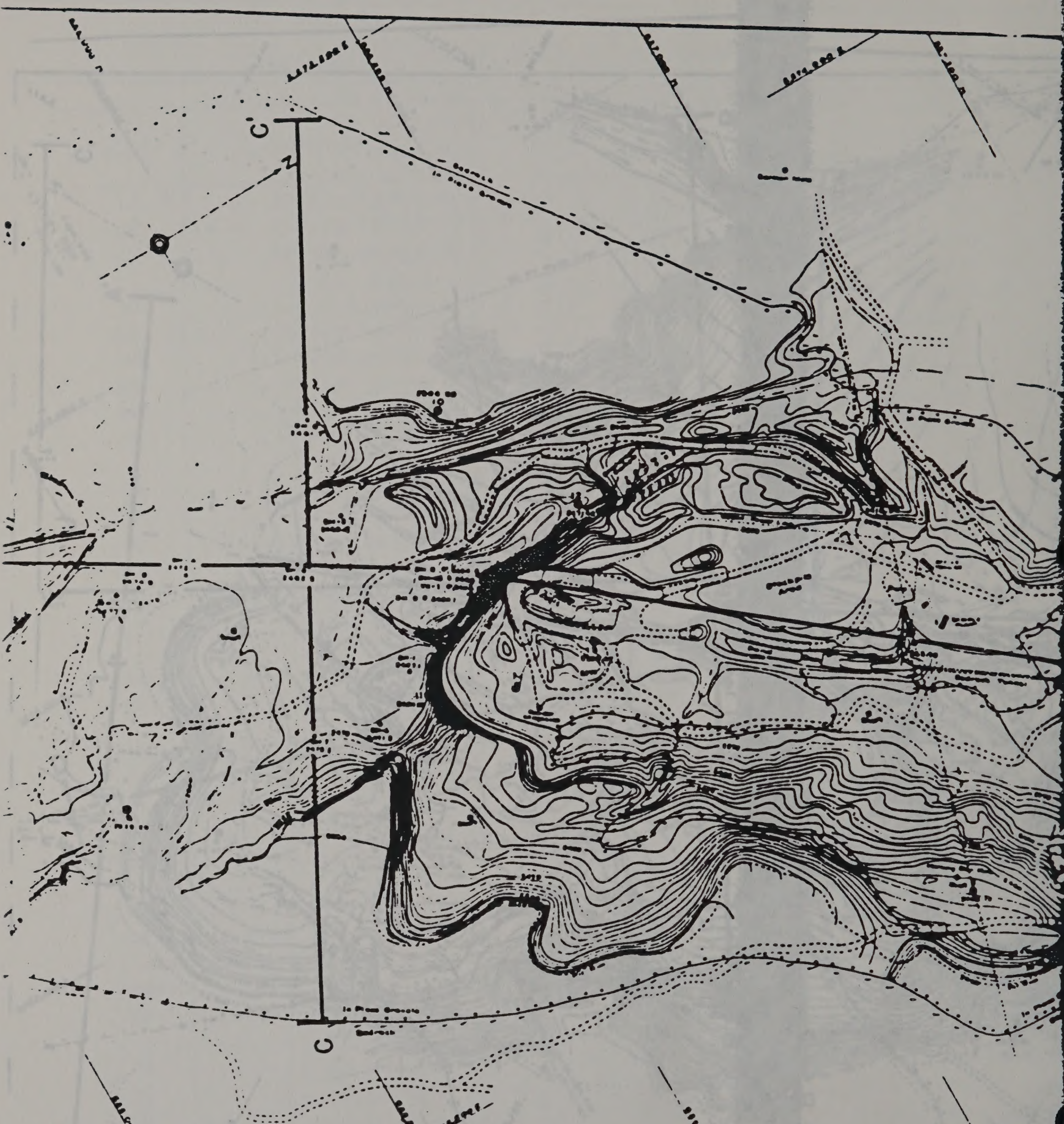
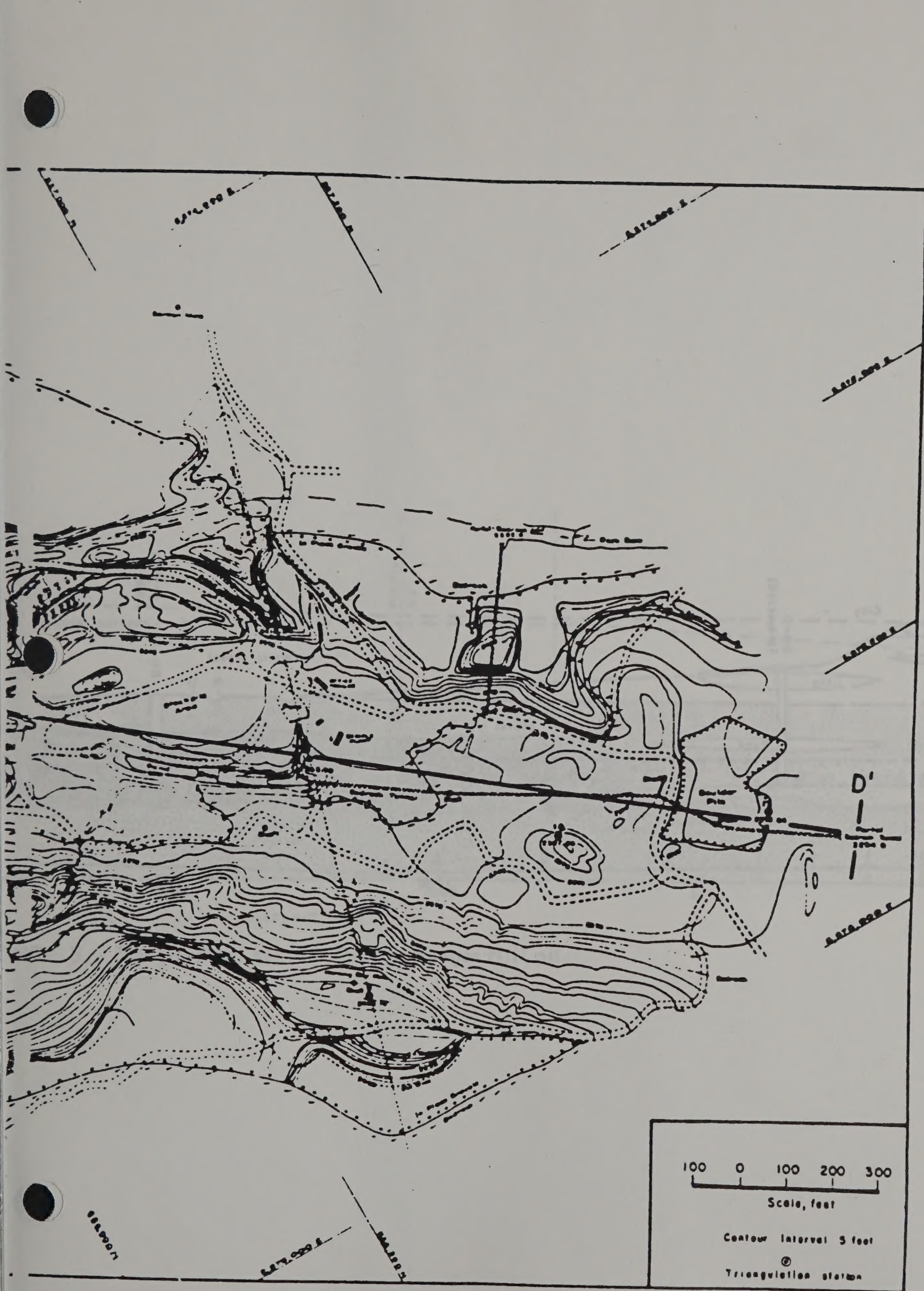


FIGURE 3. - Topographic map of Badger Hill hydrauli

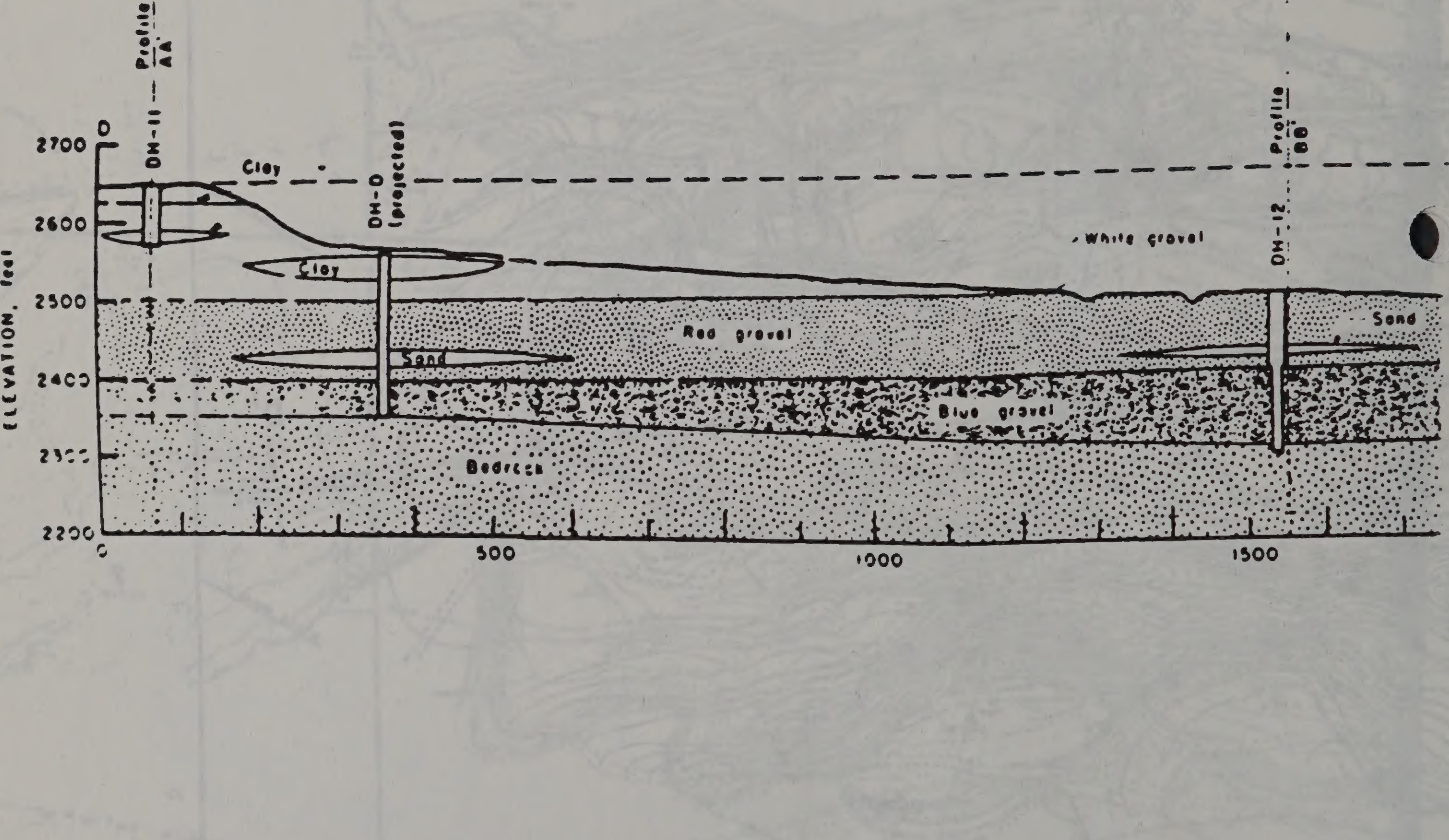
FIGURE 3. - Topographic m



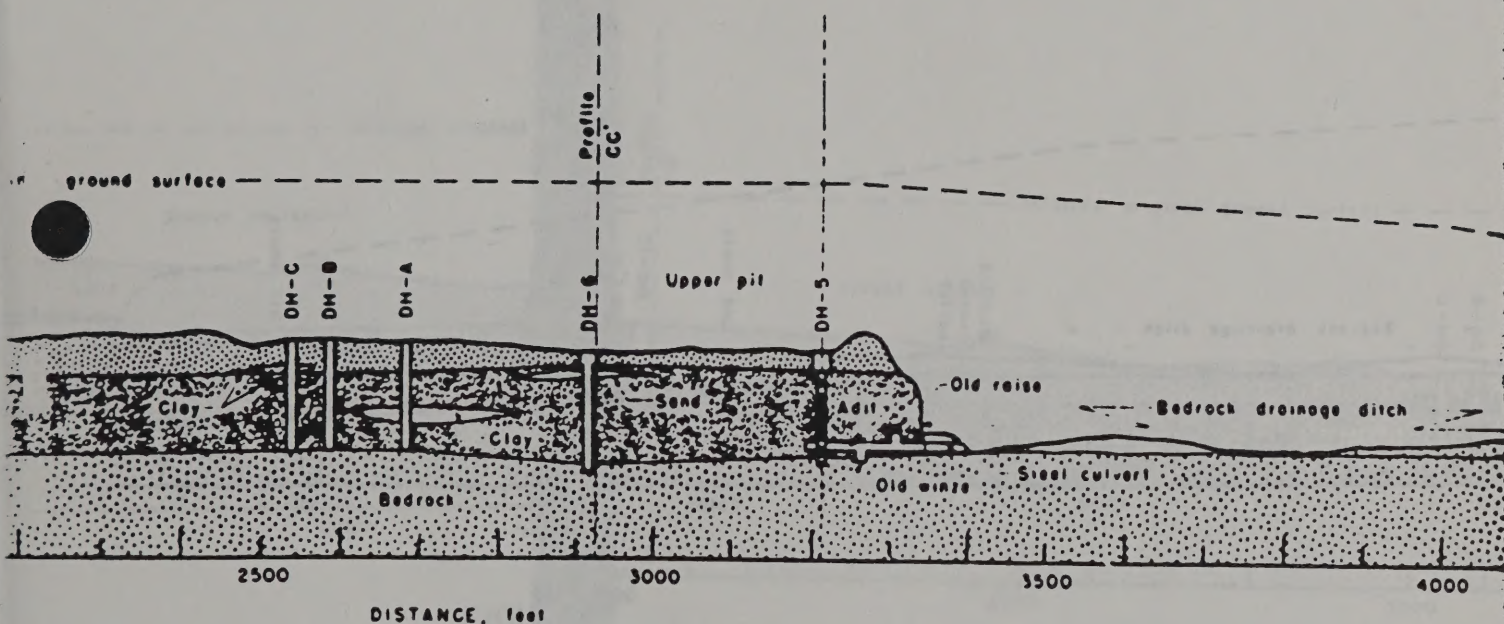
nap of Badger Hill hydraulic pit with drill hole sites.



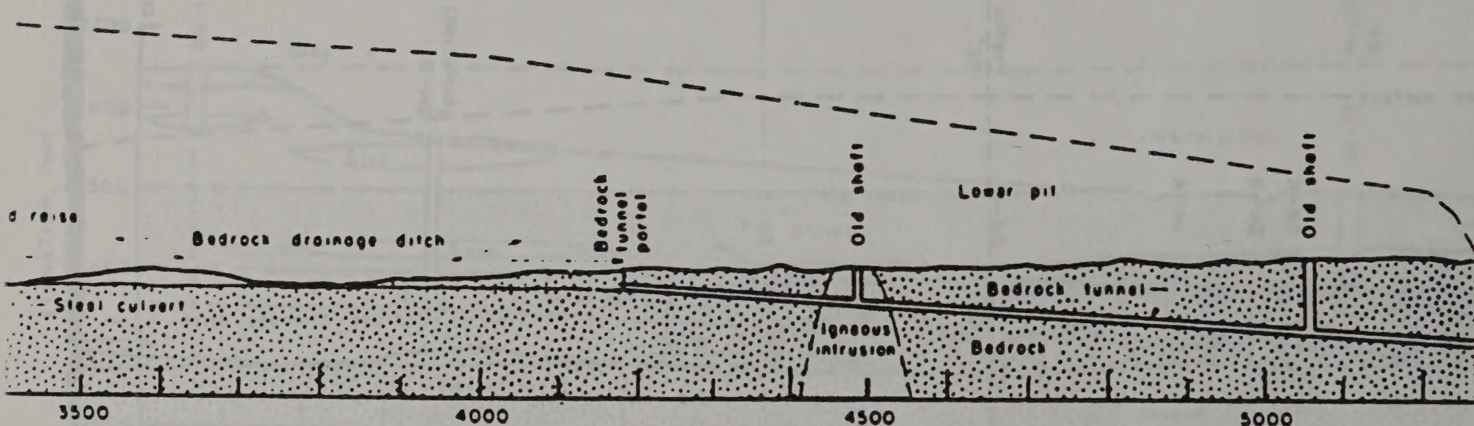
D'
100 0 100 200 300
Scale, feet
Contour interval 5 feet
Triangulation station



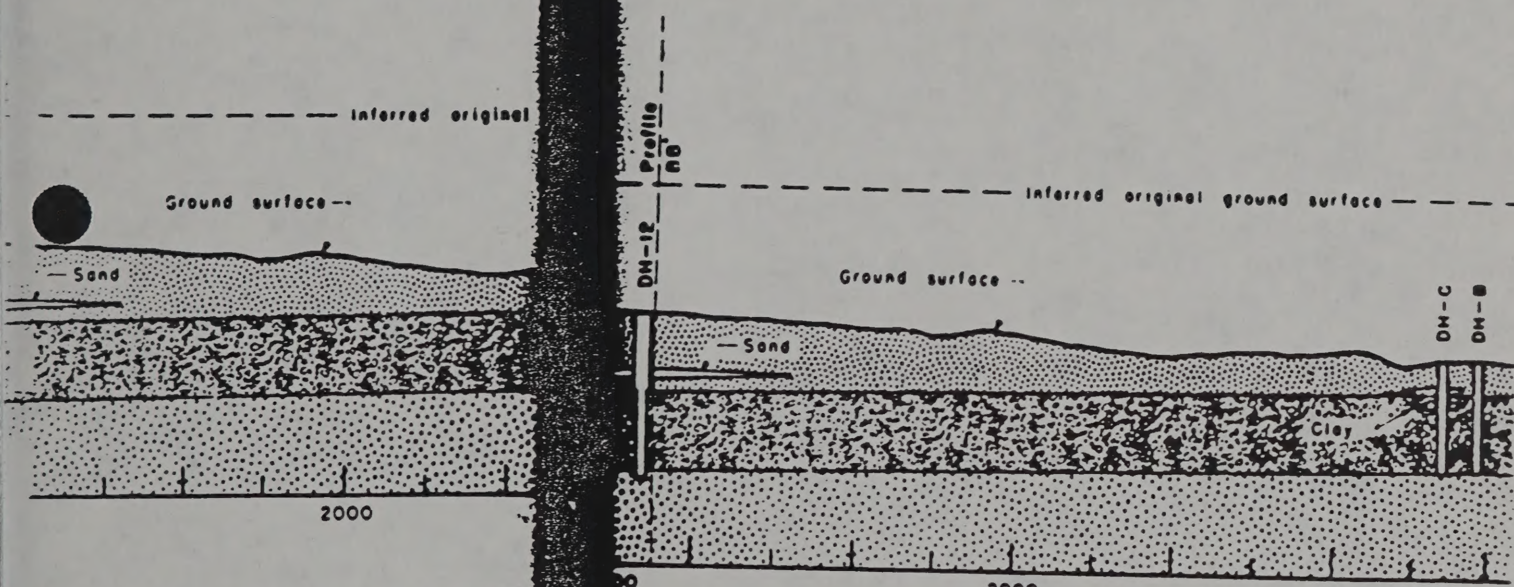
Profile AA'
DH-11
Profile BB'
DH-12
DH-O (projected)
Clay
White gravel
Clay
Red gravel
Sand
Sand
Blue gravel
Bedrock
ELEVATION, feet
2700
2600
2500
2400
2300
2200
0
500
1000
1500



GIIRE 5. - Longitudinal section through Badger Hill pit.



d raise
Bedrock drainage ditch
Bedrock tunnel portal
Old shaft
Lower pit
Old shaft
Steel culvert
Bedrock tunnel
Igneous intrusion
Bedrock
3500
4000
4500
5000



FIGU

FIGURE 5. - Longitudinal

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Geology of Placer
Deposits. (CDMG SP 34)

GEOLOGY OF PLACER DEPOSITS

Coyote diggings

Bureau of Land Management
National Training Center
9828 N. 31st Ave.
Phoenix, AZ 85051

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GEOLOGY OF PLACER DEPOSITS

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Chief, Division of Mines and Geology, Retired

ABSTRACT

The exploration of placers is a problem involving nearly all phases of the science of geology, especially physiography and stream sedimentation, neither of which has been given sufficient consideration in connection with the economic problems concerned. The use of aerial photography is a great aid in the study of placers, both ancient and modern. The use of geophysics, when applied as a part of a geologic program of exploration, can materially assist in guiding drilling operations and underground prospecting, with the result that the cost of expensive development work can be greatly reduced and hastened to an earlier completion.

Placers are classified according to the way they are formed: residual, eluvial, stream, glacial-stream, bajada, eolian, and beach. Since the ordinary stream placer is by far the most important, various phases of stream study are discussed in this paper. A need for further scientific study and the development of systematic working criteria is apparent.

The depleted placers of California consist largely of Recent and Pleistocene stream gravels and uncovered or buried, but easily accessible, Tertiary channels, while the large reserves lie in more remote positions. These are exemplified by hidden buried bench gravels not connected with the surface nor with channels already worked, and by the lower untouched channels that lie on true bedrock beneath the 'false bedrock' of the dredged areas along the western foot of the Sierra Nevada. Some Pleistocene gravels still lie in pockets beneath the waters of the larger rivers where faults have caused down-dropping of the stream bed. Benches still lie in isolated regions such as the Klamath Mountains. The desert affords several types of deposits: stream placers buried by alluvial fans, reworked older placers, gravels interbedded with lavas, and the more recent bajada placers. Marine placers of Cretaceous, Eocene, Pleistocene, and Recent periods exist in the state, which may in places be worth investigating.

The largest of all these possible reserves in California probably lies in the remaining buried Tertiary stream channels of the Sierra Nevada.

INTRODUCTION

The gold-bearing gravel deposits or *placers* of California that still remain untouched lie, for the most part, in more obscure positions than the depleted gravels which formerly produced vast wealth for the state. The depleted gravels, which were once readily accessible but are now nearly mined out, fall into three principal classes:

Depleted Placers

1. Recent and Quaternary stream gravels.
2. Uncovered channels of Tertiary age.
3. Buried Tertiary channels, easily located.

About one billion dollars worth of gold came from these three sources, the first having produced twice as much as the other two put together.

The placers which still remain to be sought out and worked, offer a challenge to the ingenuity of the exploration geologist. The problems involve the following types of deposits:

Placer Reserves

1. Deep gravel deposits lying immediately beneath several large rivers, such as the Feather and Klamath.
2. Isolated high benches such as those found in the Klamath Mountains.
3. Ancient gravels that lie beneath 'false bedrock' (interbedded volcanic layers) of the dredging areas along the western foot of the Sierra Nevada.
4. Gold-bearing gravels occurring in the 'shore' deposits of the Ione (Eocene) formation, and the Chico (Cretaceous) formation.
5. Buried Tertiary channels and associated benches located in the known gold-bearing districts of the state.
6. Buried Tertiary channels and associated benches in the lava-covered district between the Sierra Nevada and the Klamath Mountains.
7. Bajada placers, or desert alluvial fan deposits, where gold is derived directly from the original mineralized bedrock source.
8. Desert placers, where the gold is reconcentrated from more ancient gold-bearing streams.
9. Buried desert stream placers.

The scope of these problems indicates the great need of an understanding of the geological principles involved. In no kind of mining is geology more applicable than in the exploitation of these more obscure placer deposits.

Significance of Improved Exploration Methods

A widespread geologic study of the ancient Tertiary gold-bearing stream channels of the Sierra Nevada, the gravel deposits of which are found to a

large extent buried beneath a mantle of volcanic materials, was concluded by the United States Geological Survey over half a century ago. Lindgren's "Tertiary Gravels" summed up, in a splendid manner, in 1911, these various geologic studies. His data were drawn from his own careful observations, from those of his associates, H.W. Turner, F.L. Ransome, and J.S. Diller, and from such early sources as J.D. Whitney, W.H. Storms, and Ross E. Browne.

Lindgren's Colfax folio, published in 1900, was the last great detailed field study of this kind in the Sierra Nevada. By no means, however, is this folio confined to the subject of stream channels, for it deals with every phase of the geology of the quadrangle. Everything of importance which it was possible to accommodate on a map of the small scale used—two miles to the inch—was recorded. At the time this field work was done, the best equipment and finest techniques of the day were employed, and very little escaped Lindgren's keen observation, each feature being scrutinized and shrewdly interpreted by his masterful mind.

Since then, however, considerable advance has been made in exploration techniques; other and different points of vantage are now available; and a greater degree of refinement of study is therefore in order. Furthermore, mining itself has many advantages today over the earlier methods.

I. Aerial Photography

From the air, regional photographs are systematically taken by qualified aerial photographers. The pictures are then examined under the stereoscope, or used in constructing topographic maps which show the most amazing completeness of detail. Many surface features never before realized are thus simply unfolded before the eye. Geologic truths in great numbers are revealed, and many important problems solve themselves. Used as a base for location of field observations and surface mapping, these aerial photographs are unexcelled. They are undoubtedly the greatest practical aid which has yet reached the hands of the geologist; besides they give secrecy, speed, and low-cost surveying to the program of modern exploration.

II. Geophysical Surveying

Added to this regional view from the air is the greater insight into the very interior of the earth itself afforded by several types of geophysical instruments, now well-tried and standardized. Peculiar characters of rock structure and composition are not only revealed but measured with precision by skillful engineers. Since the proper interpretation of all results thus obtained requires sound geologic reasoning, it is important that a better and more detailed background of geology should be drawn, and this is made more effective by aerial photography.

III. Physiography

The subject of physiographic geology or geomorphology has in recent years made notable progress in developing sound, scientific principles concerning the history and origin of the present surface configuration of the earth. Since these principles are directly applicable to the more ancient earth surfaces of the Sierra Nevada, over which flowed the Tertiary streams now extinct, *Tertiary physiography is the key to the ancient channel problem.*

IV. Study of Desert Processes

Study of the geologic processes at work in the desert has led to a better understanding of the desert placers, which offer a practically virgin field for exploration, holding a potential wealth not yet known.

V. Stream Sedimentation

Furthermore, the study of sedimentation has now reached a refined stage of development. There are today available various methods of technique which may be extensively applied to stream deposition. This sort of research includes the critical study of texture and structure of strata, as well as the microscopic examination of their mineral grains. It should yield a wealth of practical information concerning the processes involved in the accumulation of gravels, in the nature and direction of stream flow, in knowledge of what to expect as regards the concentration of gold and other heavy minerals, and in the correlation of channels of the same period or of the same system.

The more obvious criteria of stream sedimentation have long been used by the experienced miner, who, by examining the gold particles under the simple hand lens, infers whether they were robbed from earlier channels or whether they came directly from a vein. Also he uses the 'shingling' of gravels to tell him the direction of stream flow. These and a few other working criteria now used by the miner, however, have not all undergone a thorough scientific test, and at present there is a wide difference of opinion as to their interpretation.

The advances in technique and sound geologic interpretation should, therefore, be made to serve as wide practical aids in channel exploration, and much more definite and conclusive results should be gained now than were possible years ago.

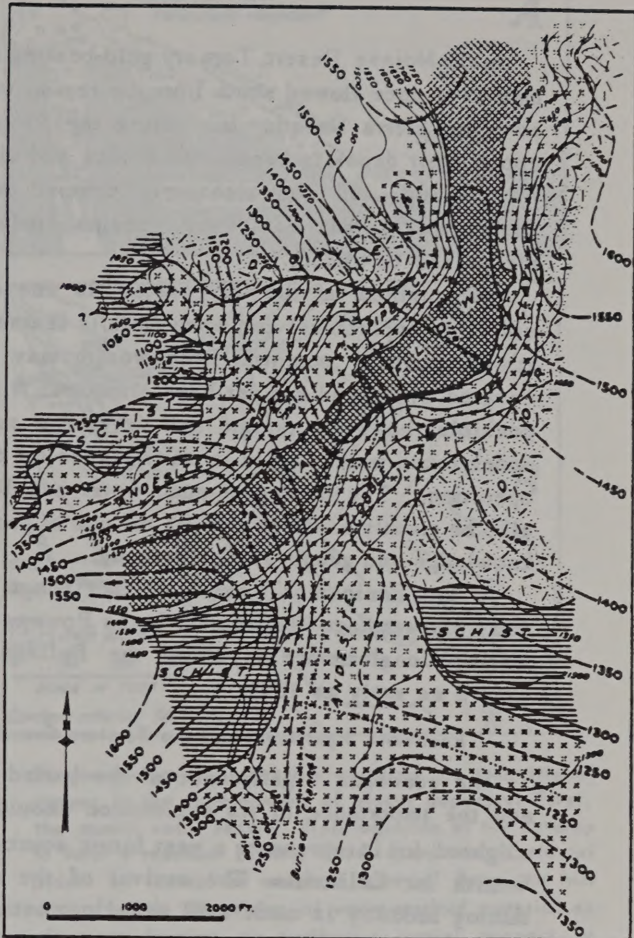
Usefulness of Contouring an Ancient Surface

The most elucidating method of tracing in detail and showing graphically the position and course of an ancient stream valley is by preparing a contour map of the old drainage surface. The contours may be superimposed over a base map which should also show the present surface topography and areal distribution of the geologic formations, as well as any mine workings, drill holes, etc.

With the old-surface contours superimposed on present surface contours, an accurate estimate may be made of the thickness, extent, and yardage of the

intervening channel-filled area. The points which are to be used in preparing an old-surface contour map should be secured through careful study of geologic and physiographic conditions, and obtained during the surface and underground survey. Drill-hole data and the essential results of geophysical observations should also be represented on this map. Careful enough study should be made so that the points used represent only one period of erosion.

If the geologic work is done prior to a contemplated geophysical survey and drilling program, much time and money may be saved in the location and number of points of observation, as well as in the number of drill holes needed. An approximate old-surface contour map may generally be constructed as a preliminary step by a skillful geologist, though he is limited only to surface data. This map should locate the general trend of the ancient valley to which later detailed work should be confined, thus eliminating much unnecessary and more costly work in adjoining areas unlikely to be productive. The most accurate elevations are those taken on bedrock where it is in direct contact with the older gravels, or with the pipe-clay and other volcanic materials of the oldest period of the area. Thus geological investigation should limit the extent of the geophysical work, which in turn defines the area to be drilled and later to be explored by underground methods.



Example of a geologic map showing the earlier prevolcanic topography in contrast with the present topography. Old-surface contours (dashed lines) are drawn over present-surface contours (fine, full lines). A major early Tertiary valley is thus reconstructed, and the general position of its buried stream channel is indicated, lying beneath volcanic materials of two later geologic periods, i.e., (1) Miocene cobble and pipe-clay (mud-flows and lake beds), (2) Pliocene lava flow which followed a canyon cut in the andesitic materials. The older channel cut in bedrock is gold-bearing. Its course was controlled by bedrock, for it followed the contact between hard diorite and softer schist. The gradient of the lava flow is not as steep where it is directed south as where it turns towards the west. This is explained by the westward tilting of the Sierra Nevada.

Provinces of the Ancient Channels in California

The so-called 'buried channels' occur for the most part in the northern Sierra Nevada, where, during the Tertiary period, millions of years ago, volcanic outbursts with their mud-flows, covered the then existing network of stream courses. Thus entrapped and preserved, this old surface of the earth, probably Eocene in age, with all its forest-covered hills, beautiful valleys, and winding streams laden with gold, became completely hidden. Not until the area was lifted by mountain-making forces did the later rivers cut their canyons through the heavy mantle and expose whatever was beneath it. But even then, further volcanic mud-flows, cobble-washes, and new streams of lavas, filled and refilled the valleys formed. Finally, modern canyon-cutting trenched the whole area deeply, leaving remnants of flat-topped ridges between.

Though no volcanic mantle ever covered the ancient streams of the Klamath Mountains, the region was uplifted in late Tertiary and Quaternary times, and the rivers were therefore entrenched. As a result, benches of gravel were left at various elevations along the sides of the canyons. Faulting also played an important role in the history of the uplift, trapping gold-bearing gravels which date back to the Eocene. The whole subject of the surface features of the region, past and present, represents a series of very interesting problems not yet entirely solved.

In the Mojave Desert, Tertiary gold-bearing streams may have once flowed south from the region which is now the Sierra Nevada; but during the Pleistocene epoch their deposits became so broken and disrupted by faulting, and so extensively covered by later desert wash, that now there remains little to be recognized or traced.

The problem of the 'dry placer' is one of considerable importance, and when more is learned about it, an immense potential gold wealth may be discovered which has not yet been glimpsed. It is quite possible, however, to find and develop a sufficient underground water supply in many places for dredging operations. The finding of such water supplies may also be greatly assisted through the use of geological knowledge and geophysical surveying.

In the Peninsular Ranges of San Diego County, some placers have been mined. In the Poway (Eocene) marine conglomerate is found the Ballena placer, known as early as 1893.

Economic Significance of the Tertiary Gravels

The economic significance of the buried channel and the importance of its exploration should not be slighted, for it represents a vast future source of gold wealth for California. The revival of the channel-mining industry is made still more interesting by an understanding of the detailed geologic features, the possibilities they suggest for certain of these buried stream courses, and the realization of the vast extent of the region yet to be explored.

Since an estimate of the potential wealth of the desert placers depends upon further exploration and mining of them, it is not yet possible to evaluate their place in this study.

History of Development

The discovery of the Tertiary channels followed shortly after the discovery of gold in the present stream courses. In places, remnants of Tertiary channels were found lying exposed high up on 'flats,' stripped of their volcanic covering by Pleistocene erosion. Starting from these flats, the miners followed the uncovered channel to the point where it was covered by the lava capping, the top of which formed another kind of 'flat.' Water was nearly always encountered, which led to the construction of long and expensive drainage tunnels. To place these tunnels at the proper elevation to serve their best purpose was the most serious problem, for the old-timers did not have the powerful pumps which we use today.

The ideas developed concerning the courses and positions of the ancient channels were many and varied. Misconceptions of Tertiary physiography led many persons astray, and millions of dollars were spent in vain. In spite of the millions gained, the losses were so great as to stamp this form of mining as hazardous in the extreme.

It is most instructive to follow carefully the recorded history of mining in a given district and to consider its relation to the geologic condition of that area. The two are so closely related as to provide a guide to the probable history which might be expected to be found in another area if the geology were known, or *vice versa;* the recorded history reflects what geologic conditions may be expected.

The geologic history and structure of the buried channels are so complex that the best of engineers have been baffled by them. Fragmentary benches and segments of rich gravel deposits which still rest in positions completely hidden from the surface, or even from the underground passages which enter into the lower main channels afford alluring possibilities to the geologist and geophysicist as well as to the prospector. A three-dimensional surface, complex and irregular in the extreme, is the problem to be faced.

The key to the solution is geology, aided by aerial photography, followed by geophysical surveying, and finally by directed prospecting through means of the drill, shaft, incline, or tunnel. To be effective, all these methods should be coordinated into one unified exploration program.

CLASSIFICATION OF PLACERS

Outline of Classification

A systematic geological study of placers calls for an orderly classification dividing them into *genetic types,* which indicate how they were first formed. The following classification is based upon the fundamental conditions of deposition:

Fundamental Classification of Placers

A. Residual placers or 'seam diggings.'
B. Eluvial or 'hillside' placers, representing transitional 'creep' from residual deposits to stream gravels.
C. Bajada placers, a name applied to a certain peculiar type of 'desert' or 'dry' placer.
D. *Stream placers (alluvial deposits), sorted and re-sorted, simple and coalescing.*
E. Glacial-stream placers, gravel deposits transitional from moraines, for the most part valueless.
F. Eolian placers, or local concentrations caused the removal of lighter materials by the wind.
G. Marine or 'beach' placers.

Of these seven types, the ordinary stream placer is by far the most improtant. All types are, however, more or less interrelated and intergradational; they are all subject to deformation or burial, and they may be formed during any geological period.

Most of these various types of placer deposits are well known in Alaska. B.N. Webber proposed the name, "Bajada placer" for peculiar desert, or so-called 'dry' placers, which he carefully analyzed (*American Institute of Mining and Metallurgical Engineers* Technical Publication 488, 1935). The concentration of gold by the agency of wind in Western Australia has been described by T.A. Rickard and Herbert C. Hoover (*American Institute of Mining and Metallurgical Engineers* Transactions, vol. 28, 1898, pp. 490-537; 758-765).

General Statement

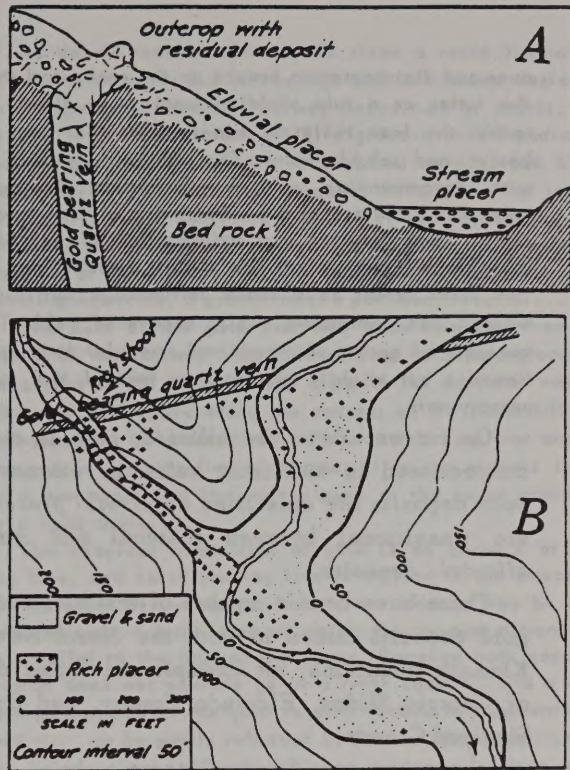
In valuating a placer, one of the first considerations should be the determination of how it was formed. It is recommended, therefore, that the exploration engineer should classify genetically each gravel deposit to be prospected. This calls for an understanding of the historical geology of the region and of the processes which have been responsible for the formation of the deposit, as well as how it came to be preserved or modified from its original form. The actual sampling of a deposit is carried on in a much more intelligent and satisfactory manner when a clear understanding of the geologic set-up has been acquired.

CHARACTERISTICS OF THE PRINCIPAL TYPES OF PLACERS

Residual Placers

In order that gold may become released from its original source in bedrock, the encasing material must be broken down. This is most effectively done by long-continued surface weathering. Disintegration is accomplished by persistent and powerful geologic agents, which effect the mechanical breaking-down of the rock and the chemical decay of the minerals.

The surface portion of a gold-bearing orebody will become enriched during this process of rock disintegration, because some of the softer and more soluble parts of the rock are carried away by erosion, leaving the remaining portion of higher tenor. The name *residual placer* is applied to this type of deposit. After the residual portion is mined away by comparatively inexpensive methods, the harder mineralized rock is encountered, and the mining methods must be changed to accommodate another type of deposit, i.e., the lode.



A. Diagrammatic cross-section showing the transitional stages in the development of placer deposits: First, the quartz vein; second, disintegration at the outcrop to form a *residual* placer; third, formation of *eluvial* placer by 'creep' of residual material down the hill slope; fourth, deposition of water-worked material as *alluvium*, forming an auriferous gravel deposit, or stream placer.

B. Sketch map showing the development of rich placers broken-down directly from the disintegration of a gold-bearing vein. (*After Lindgren, "Mineral Deposits."*)

The so-called 'seam diggings' are in weathered, gold-bearing quartz stringers, occurring along fracture zones of disintegrated schists.

Eluvial Placers

After gold is released from its original bedrock encasement through agents of rock decay and weathering, the whole weathered mass may 'creep' down the hillside (in some regions partly because of frost heaving) and may finally be washed down rivulets into gulleys. Lindgren in *Mineral Deposits* (p. 213), stated:

> When the outcrops of gold-bearing veins are decomposed a gradual concentration of the gold follows, either directly over the primary deposits or on the gentle slopes immediately below. The vein when located on a hillside bends

over and disintegration breaks up the rocks and the quartz, the latter as a rule yielding much more slowly than the rocks: the less resistant minerals weather into limonite, kaolin, and soluble salts. The volume is greatly reduced with accompanying gold concentration. The auriferous sulphides yield native gold, hydroxide of iron, and soluble salts. Some solution and redeposition of gold doubtless take place whenever the solutions contain free chlorine. The final result is a loose ferruginous detritus, easily washed and containing easily recovered gold. This gold consists of grains of rough and irregular form and has a fineness but slightly greater than that of the gold in the primary vein.

On its way down the hillside, gold is sometimes concentrated in sufficient value to warrant mining. Such deposits are classified as *eluvial placers.* They are transitional between residual and stream, or *alluvial,* deposits.

There have been a number of residual and eluvial gold deposits mined in both the Sierra Nevada and Klamath Mountains; for example the 'seam diggings' of Georgia Slides, Eldorado County, and Scott Bar, Siskiyou County.

Stream Placers

By far the most important type of placer is the ordinary alluvial gravel or stream placer. So far, it has been the source of most of the placer gold mined; but now its supply is nearing depletion, save for values remaining in those ancient channels which lie deeply buried beneath a cover of lava or rock debris.

Deposits by streams include those of both present and ancient times, whether they form well-defined channels or are left merely as benches. Stream placers consist of sands and gravels sorted by the action of running water. If they have undergone two or more periods of erosion, and have been re-sorted, the result will in all probability be a comparatively high degree of concentration of the heavier mineral grains.

Quoting from J.B. Mertie (U.S. Geological Survey Bulletin 739):

All bench placers, when first laid down, were stream placers similar to those of the present stream valleys. In the course of time the stream gravels, if not reworked by later erosion, may be left as terraces or benches on the sides of the valley, if the local base-level is lowered and the stream continues to cut down its channel. Such deposits constitute the so-called bench gravels. On the other hand, if the regional or local base-level is raised, the original placer may be deeply buried and a second or later placer deposit may be laid down above it.... If the local base-level remains practically stationary for a very long period, a condition seldom realized, ancient and recent placers may form a perfectly continuous deposit in a long valley, for the deposition of a gold placer is known to occur at that point in a valley where the stream action changes from erosion to alluviation, and such deposits are therefore formed progressively upstream.

Where several parallel and contiguous streams that are forming placers emerge from their valleys upon an open plain, perhaps into some wide valley floor, a continuous or coalescing placer may be formed along the front of the hills. If the streams empty into some lake or estuary, a delta placer, genetically the same but perhaps different in some minor respects, may be formed. Manifestly such compound placers may be formed by either present or ancient streams and may be elevated or buried in the same way as simple stream placers.

In order to understand thoroughly the subject of stream placers, streams themselves must be studied in regard to their habit, history, and character. The effects of existing and changing climates, the relation to surrounding geologic conditions, and the effect of movements of the earth must also be considered.

Glacial-stream Placers

It is a frequent fallacy of the placer miner to attribute the deposition of gold-bearing gravels to the action of glaciers. Contrary to such a belief, glaciers do not concentrate minerals; the streams issuing from melting ice, however, may be effective enough in sorting debris to cause placers to be formed under certain especially favorable conditions. In California, glaciers occurred throughout the high Sierra during the Pleistocene, but in Tertiary times they were wholly lacking. The Pleistocene streams cut through the earlier channels, robbing them of much of their gold.

Blackwelder has said (28th Report of the State Mineralogist, p. 309-10):

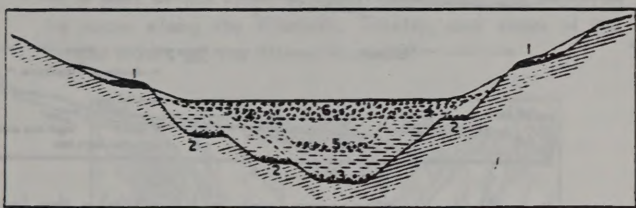
Since it is the habit of a glacier to scrape off loose debris and soil but not to sort it at all, ice is wholly ineffective as an agency of concentration for metals. Gold derived from the outcrops of small veins is thus mixed with large masses of barren earth. Attempts to mine gold in glacial moraines, where bits of rich but widely scattered float have been found, are for that reason foredoomed to failure.

If a glacier advances down a valley which already contains gold-bearing river gravel, it is apt to gouge out the entire mass, mix it with much other debris and deposit it later as useless till. Under some circumstances, however, it merely slides over the gravel and buries it with till without disturbing it.

On the other hand, the streams born of glaciers or slowly consuming their moraines have the power to winnow the particles of rock and mineral matter according to size and heaviness. Such streams may form gold placer deposits in the well-known way by churning the load they carry and allowing the heavy minerals to sink to the bedrock. Placers may therefore be found in the deposits of glacial rivers if there are gold veins exposed in the glaciated area of the stream. Nearly all the gravel which has been dredged for gold along the foothills of the Sierra Nevada was deposited

by rivers derived in part from glaciers along the crest of the range, but most of the gold was probably picked up in the lower courses of such rivers. Since glacial rivers choke themselves and build up their channels progressively, their deposits are likely to be thicker and not so well concentrated as those of the more normal graded rivers which are not associated with glaciers.

A few gold-bearing deposits in re-worked glacial till may be found along the eastern front of the Sierra Nevada, as, for example, in the region just north of Mono Lake.



Cross-section of a gold-bearing desert stream valley (Manhattan, Nevada), showing the results of several periods of stream deposition from the oldest (1) to the youngest (6). (*After Ferguson, United States Geological Survey, Bull. 723.*)

Bajada Placers

A bajada is a confluent alluvial fan along the base of a mountain range. B.N. Webber described the *bajada type* of placer deposit as follows:

Bajada is the Spanish term for slope and is used locally in the Southwest to indicate the lower slope of a mountain range, the portion consisting of rock debris and standing at a much lower angle than the rock slope of the range proper....

The total production of gold from bajada placers in the southwestern United States is necessarily small, probably not over ten million dollars....

Most of all bajada placer gravels are Quaternary and the larger part are Recent...

The genesis of a bajada placer is basically similar to that of a stream placer except as it is conditioned by the climate and topography of the arid region in which the placer occurs....

Erosion, transportation and deposition in a region of extreme aridity present some phenomena not encountered in more humid areas. Practically all the work of running water is strongly conditioned by aridity....

Rock-floored canyons through which rock fragments are moved by infrequent torrential floods should constitute excellent pebble mills for the further reduction of the material, but the amount of attrition accomplished seems to be slight, as fragments, large or small, on the bajada slope are decidedly angular and show little effect of attrition.... Probably a small percentage of the gold is freed during this phase of the movement of gravel. The gradient of these intermont drainage channels is too high to permit lodgment of the finer gravel. When a small amount of gravel is temporarily lodged in one of these channels, the deposit displays most of the characteristics of stream gravel.

As debris reaches the bajada slope a rapid diminution in volume of water due to seepage and an extreme decrease in the grade of channel causes deposition of debris, and either (1) an alluvial fan or (2) a gravel-mantled pediment may be formed. If detritus is supplied to a bajada slope much faster than it can be removed, an alluvial fan is the result.... If rock debris is supplied to the bajada slope in considerable volume but not in excess of the quantity capable of transference to the center of the basin by the existing agencies, a gravel-mantle pediment results....

The bulk of the gold that has been released from its matrix on the journey from lode outcrop to bajada slope is deposited on the bajada slope close to the mountain range. The gold is dropped along the contact of the basin fill and bedrock; this is referred to hereafter as the lag line and is coincident with the line of contact of bajada gravels lying at a low angle and the rock slopes of the range standing at a high angle....

The heaviest deposition of gold is on bedrock at the lag line, and since the lag line is moving in the direction of the crest of the range, values on bedrock may be distributed over a large area of which the longest dimension is parallel to the foot of the range. Because bulk concentration does not operate as in a river channel, and a certain percentage of the gold is still locked in fragments of matrix, to be partly released by further disintegration on the bajada slope, there is a strong tendency for less gold to reach bedrock and for more to remain erratically distributed throughout the detritus than in the case of stream gravels.

Eolian Placers

Bajada placers usually show enrichment on the surface due to removal of lighter material by wind and sheet floods. This is true of some of the dry placers of California, though no commercial eolian gold deposits such as those mined in Australia are known in this state.

Wind action, however, is responsible for the removal of large amounts of fine detritus in the desert. The process involved has been called *deflation*. It is quite likely that it will be found to play an important part in the surface concentration of desert placers.

Spurr described "auriferous sand dunes" in the Nevada desert seven miles south of Silver Peak, 18 miles from the California boundary line, in his *Ore deposits of the Silver Peak quadrangle* (U.S. Geological Survey Professional Paper 55).

Beach Placers

Concentrations of heavy minerals occur in various places along the Pacific Coast as a result of the action of shore currents and waves, which tend to sort and distribute the materials broken down from the sea cliffs or washed into the sea by streams. The

heavy minerals consist for the most part of magnetite, chromite, ilmenite, monazite, and zircon, with occasional fine particles of gold and platinum. Beach placers are of two kinds, (a) present beaches and (b) ancient beaches. An elevated coast line is often found overlaid with terrace gravels which were deposited at a time when the coastline was depressed. The beach placers of economic importance are those that have been reconcentrated over and over again.

Excellent descriptions of the geologic processes involved may be found in reports on beach placers of Nome, Alaska, and of the coast of Oregon and California. In discussing the origin of the gold in the Oregon beach placers, J.T. Pardee says (U.S. Geological Survey, Circular 8):

> Some of the miners believe that the gold of the beaches comes up out of the sea, an idea suggested by the fact that after a storm a formerly barren stretch may be found to be gold-bearing. This notion is true so far as the immediate source of some of the gold is concerned. Materials composing the foreshore are carried out in the offshore zone at one time and returned to the beach at another. In the process a shift up or down the coast may occur.... Soundings of the Coast and Geodetic Survey show black sand to occur in the offshore zone at the present time. Gold and other minerals are doubtless present also.... For the beaches that border retreating shores, however, the most of the gold and other minerals come directly from the rocks that are being eroded by the waves.

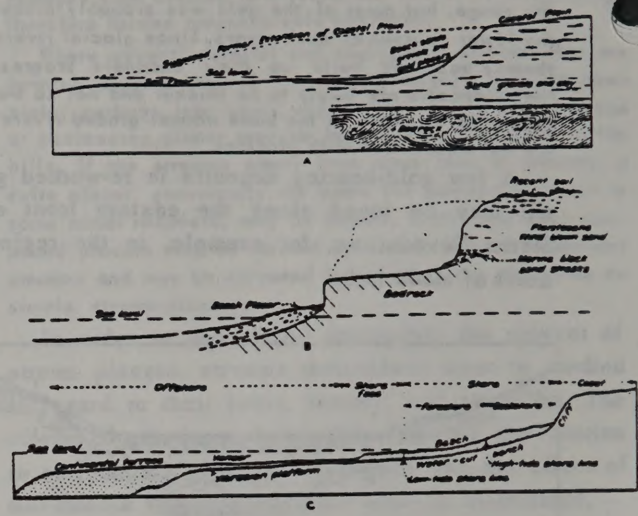
The economic possibilities of mining the black sands of the California coast for their gold content have long been discussed.

Gold-bearing gravels of marine origin occur in the Chico (Upper Cretaceous) sediments of northern California. That they are marine in origin and not fluvial is shown by their content of abundant fossil seashells, as well as by the character of their strata. They were formerly wrongly classed as "the gravel-filled channel of a Mesozoic river." Gold-bearing gravels have also been reported from marine sediments of the Lower Cretaceous of northern California.

Since the gravels of the Eocene rivers of the Sierra Nevada were richly gold-bearing, it is to be expected that some of the gold reached the sea. The sedimentary deposits of this Eocene sea are known as the Ione formation. They occur along the western foot of the Sierra Nevada.

Lindgren says:

> At the mouth of the rivers which descended from the Tertiary Sierra Nevada extensive delta deposits were accumulated, and it is thus difficult in many places to draw any exact line between the Ione formation and the river gravels proper. The gravels in the formation are



A. *Diagrammatic cross-section illustrating the formation of beach placers in Alaska. (After Collier and Hess, United States Geological Survey, Bull. 328.)*

B. *Cross-section of a typical beach placer (Oregon). (After Pardee, United States Geological Survey, Circular 8.)*

C. *Diagrammatic cross-section of a coast, showing shore zones in an advanced stage of development. (After Johnson, see Pardee, United States Geological Survey, Circular 8, 1934.)*

> locally auriferous, though generally poor, because spread over large areas.

The deposits contain quartz gravels and finely divided quartz grains; they are closely connected with the oldest river-channel deposits; they occur along the extreme western foot of the Sierra Nevada. Therefore the Ione sediments

> ... indicate delta deposits formed at the mouths of many westward-flowing streams. Marine fossils in the upper part of the Ione formation show that they accumulated on the shores of an Eocene sea.

The processes involved in the distribution and concentration of gold in the marine strata of both the Cretaceous and Eocene formations have never been carefully studied. If these marine and delta placer deposits have any particular economic significance, it certainly has not been adequately demonstrated.

PRESERVATION OF PLACERS

Placers are preserved if something keeps them from being eroded away. Since streams are continually changing their positions, fragments of their deposits are often left isolated. In cutting a deeper channel, a stream leaves 'benches' or 'terraces' at different intervals along its valley sides; but erosion tends to destroy them, unless they are protected in some way.

Burial is the most effective way in which a placer may be preserved. The name 'buried channel' has often been restricted to streams covered deeply by lavas, mud-flows, ash-falls, etc., all of which were very common during the Tertiary period in the Sierra Nevada. There are, however, other means by which burial may be effected.

1. By covering with landslide material. (An example occurs in Canyon Creek, Trinity County.)
2. By covering with gravel, caused by the faulting-down of a part of the river system. (Examples are believed to occur along the Klamath, Trinity, and some of the larger rivers of the Sierra Nevada.)

Ideal sketch showing how a landslide may dam up a mountain valley to form a lake. Silt, sand, and gravels deposited in and on the edge of this lake will cover the stream gravels in its bottom. *(After Davis, State Mineralogist's Report XXIX).*

3. By covering with lake deposits. (Many of the buried Tertiary channels were covered first by lake sediments called 'pipe-clay', before lava or mudflows poured over them.)
4. By covering with gravels when the stream is choked. (Examples are common along streqm systems.)
5. By covering with gravel when the stream course is lowered below the general base-level of erosion. (Examples of this case are found along the western foot of the Sierra Nevada.)
6. By covering of the bedrock surface of down-faulted blocks, *graben*, by sediments of various sorts. (Many examples, especially in the Great Basin and Mojave Desert regions.)

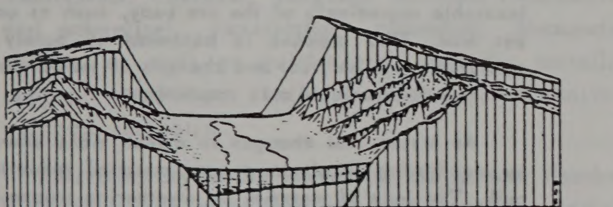


Diagram showing a down-dropped fault block *(graben)* between two uplifted fault blocks. Erosion covers the one with materials derived from the others. Streams cut in bedrock prior to faulting may thus be buried under the alluvium of the graben. *(After Davis, State Mineralogist's Report XXIX.)*

7. By covering of older stream courses with alluvial fan material, as conditions favorable to stream existence fail. (Many examples in the Great Basin and Mojave Desert regions.)
8. By covering with glacial till. (Examples may be looked for in the glaciated areas of the Sierra Nevada.)
9. By covering of beach placers with marine sediments as the fluctuating coast is submerged, but later elevated. (Such as the present elevated beach placers which are in places covered with other marine sediments.)
10. By covering of one geologic formation with another, through the processes of earth deformation and thrust faulting. (In a geologically active region such as California, examples of this case might very well be found.)
11. By submergence of river canyons to great depth beneath the ocean. (Off the coast of California many submarine channels have been discovered and mapped by the U.S. Coast and Geodetic Survey.)

To find gold-bearing stream channels, buried and preserved in such a manner that they may be profitably mined, is the challenge to the exploration geologist.

MODIFICATION OF PLACER DEPOSITS

Placer deposits may be greatly modified in form and structure by earth deformation. The gravel content may also become firmly cemented by interstitial deposition of mineral matter, such as by lime and iron carbonate, or silica, through the action of infiltrating solutions. The older the placer, the more apt it is to have been modified in these ways from its original form and attitude.

The regional tilt of the Sierra Nevada has increased the gradient of the Tertiary channels (where they lie in the direction of the tilt) from 20 or 30 feet to the mile to twice, three times, or even several times that amount. Locally, tilting has been even greater. In places where the ancient channels lie in opposite direction to the tilt the original gradient may have been reversed. Often steep tilting is accompanied by local faulting of a few feet to several hundred feet. Generally the down-throw is on the east side of the fault plane. In form, this is a replica of the action which took place and still is taking place along the eastern escarpment of the Sierra Nevada. Such displacements and changes in channel-gradient as well as in actual position of the channel, are important factors which greatly influence mining procedure. They should be understood so far as surface data will permit, before actual mining is started in a given area.

In the Mojave Desert and Great Basin region, faulting and tilting have been extremely active, greatly affecting streams antedating late Tertiary and

early Quaternary periods.

The flow of ground water through stream gravels, the former channels of which have been blocked, cut off, tilted, or folded by earth movements, is a factor of considerable consequence when it comes to mining such placers.

GOLD IN PLACERS

Original Source of Gold

The particles of gold found in placers originally came from veins and other mineralized zones in bedrock, from which they were released through surface weathering and disintegration of the rock matrix. Though the original source may not in every case have been a deposit which could today be mined at a profit, the richer placers usually indicate a comparatively rich source. A long period of deep weathering, resulting in separation and release of large quantities of gold from the bedrock, followed by a more active period of erosion, generally due to uplift, is an ideal condition for gold to be swept into stream channels and there to be concentrated into rich placers. Still richer deposits may be formed through reconcentration from older gold-bearing gravels.

These are the most important geologic conditions which have been found to exist in the various gold belts of the world, and particularly in the Sierra Nevada of California.

For the most part, the original source of gold is not far from the place where it was first deposited after being carried by running water. This is certainly true in both the Sierra Nevada and Klamath Mountains. The streams, flowing through regions of metamorphic and intrusive igneous rocks threaded throughout by gold-bearing veins, were found by early miners to contain auriferous gravels. But the more recent streams which have had only barren lavas to pass over, as in the volcanic covered area between the Sierra and Klamath regions, have proved to be barren.

To quote Lindgren (*Mineral Deposits*) again:

> The great majority of gold placers have been derived from the weathering and disintegration of auriferous veins, lodes, shear zones, or more irregular replacement deposits.... In many regions the rocks contain abundant joints, seams, or small veins in which the gold has been deposited with quartz.... It is often stated that gold is distributed as fine particles in schists and massive rocks and that placer gold in certain districts is derived from this source. Most of these statements are not supported by evidence, though it is not denied that gold may in rare instances be distributed in this manner.

Release of Gold from Bedrock

Without some widespread process of release from the quartz veins and rocks — vaults in which the metal was originally firmly held — gold particles could not have escaped to be transported as such by running water. Therefore, extensive rock weathering and decay over a long period of time is a primary factor of extreme importance. It has permitted the original source to contribute gold particles, large and small, to placer accumulations. The same geologic processes which form residual and eluvial or 'hillside' concentrations of commercial merit operate in the general release of gold from bedrock.

The factors of prime importance in weathering are *solution*, changes of *temperature*, depth of *water-table* and therefore depth of *oxidation*, action of *rain*, effect of *gravity*, growth of *vegetation*, nature and *composition of material* acted upon, and degree of *topographic relief*. Rock weathering, especially *complete* disintegration down to the water-table, rather than *deep* disintegration, is often more favored by tropical climates. This, however, is only one factor, and large areas of deep secular weathering are found in the north, in such places as Alaska, where placers are abundant.

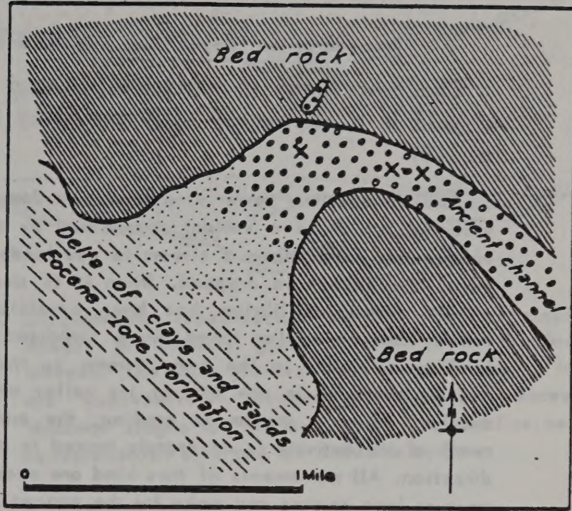
The processes which take place in the separation of gold from bedrock are described in detail by A.H. Brooks, who says, in his article on the gold placers of the Seward Peninsula (U.S. Geological Survey Bulletin 328, pp. 125-127):

> The breaking down of the rock and the accompanying chemical changes of the constituent materials set free the gold, one of the relatively indestructible minerals, and this becomes intermingled with the other insoluble material. Clay dominates in the residual mass, but if the parent rock contained quartz, this, too, usually remains, being probably the most refractory of all the common minerals toward purely chemical agencies. Mineralized vein quartz very commonly carries easily decomposed minerals, such as pyrites, and is therefore readily broken up, allowing the insoluble ingredients of the ore body, such as gold, to be set free. This process is hastened by purely physical agencies, such as frost and changes of temperature, which break up the insoluble rock constituents....
>
> As a rule, the changes in a rock mass brought about by weathering result in a very material reduction in its bulk. The loss of material by weathering among siliceous crystalline rocks, according to Merrill, amounts to more than 50 per cent, and in the purer forms of limestone it may reach as high as 99 per cent. Pumpelly [*American Journal of Science* 3rd Series, vol. 17, p. 1361] has estimated that in the limestone areas of the Ozark Mountains the residual material represents only from 2 to 9 per cent of the original rock mass. Such reductions in volume necessarily result in

more or less concentration of any insoluble material that may have been disseminated in the parent rock. This concentration will be materially greater in the case of substances of high specific gravity, such as gold, than in that of the lighter minerals, for the former will have a constant tendency to settle to the bottom of the loose material. On declivities gravity will accelerate the process and help to sort the material, producing in some places a rough stratification. This is a secular process and will proceed as long as the rocks continue to disintegrate....

It is evident that the effectiveness of all these agencies is proportional to the length of time in which they are operative. A land mass must remain stable relative to sea level for a long period of time to permit the accumulation of any considerable amount of residual material. Uplifts bring about renewed activities of the watercourses, and the residual mantle is quickly removed by erosion. It is evident that the conditions that are most favorable to the accumulation of residual material are those in which the land mass is at or near base-level when erosion is reduced to a minimum.

It seems that topographic and climatic conditions which existed during the Eocene period in the Sierra Nevada were very favorable for the release of gold. Rejuvenated drainage at the close of the period swept the immense quantities of gold, freed from the enclosing hard matrix, into the early Tertiary stream channels; and these, soon after, became buried and preserved by lake sediments, masses of gravel, cobble-wash, volcanic ash, breccia, and lava flows.



Sketch map of an early Tertiary channel and its delta in the Eocene (Ione) sea. The crosses indicate where the gold has been mined in the channel — on a bend in the stream, and at the point where a tributary entered. Finely divided gold particles occur interbedded in lenses of quartz pebbles and sand lying above clay layers ('false bedrock').

Associated Minerals

Mineral grains that are very heavy and resistant to mechanical and chemical destruction accompany the gold in placers. The so-called 'black sands,' generally made up principally of magnetite, are well known to the miner. A long list of the minerals found in sluice-box concentrates is recorded by the United States Bureau of Mines (Information Circular 6786). Besides magnetite, there are found titanium minerals (ilmenite and rutile), garnet, zircon, hematite, chromite, olivine, epidote, pyrite, monazite, limonite, platinum, osmiridium, cinnabar, tungsten minerals (wolframite and scheelite), cassiterite, corundum, diamonds, galena, as well as quicksilver, amalgam, metallic copper, bird-shot, bullets, hobnails, penknives, watches, and nails.

Buried deeply in the gravels of the modern Feather River was once found (and someone thought it was an ancient fossil) the remains of a mule's hind leg with hoof and iron shoe nailed to it. What may be found in some of the placers of today may not, therefore, be representative of what was deposited by the more ancient streams.

The presence of quantities of magnetite associated with extremely fine gold particles, makes a difficult metallurgical problem. To the geophysicist, however, the presence of any minerals having a strong effect on the magnetometer is a godsend to effective exploration.

The determination of heavy minerals and their approximate relative percentages has been extensively used in subsurface correlation of sedimentary beds in the oil fields. The same method of research could well be applied to the tracing of channels. Though it has not yet been given consideration in California, this interesting field is open for study with a well-developed technique available.

The source of the minerals deposited and associated with the gold particles lies in the rocks over which the stream has flowed. The source of chromite, platinum, and diamonds is generally attributed to belts of serpentine and related ultrabasic igneous rocks, while garnet, ilmenite and magnetite might come from metamorphic rocks, and monazite, zircon, cassiterite, wolframite, and scheelite would probably have their source in granite pegmatites.

Transportation, Deposition, and Retention

The proccesses of transportation and deposition of gold in a stream are aptly stated by Mr. Brooks (p. 128):

> The transporting power of a stream is dependent on its velocity, which is a variant determined by the gradient, volume and load. When a stream is overloaded with sediment, the excess is dropped. When it is underloaded, it erodes. When equilibrium has been established, neither erosion nor deposition takes place. Gradient, volume and load usually vary in the same stream so that deposition may be going on in one part of its valley and erosion in another. When a stream is eroding, the material within reach of its activity is constantly moved in a downstream direction. All movements of this kind are accomplished by more or less sorting and make for the concentration of the heavier particles.
>
> Deposition takes place in a stream when the velocity is decreased, either by the periodic changes in volume or by a change of gradient. Where there is a change of grade, resulting in diminished velocity, the gold is laid down with the other sediments. It must be remembered, however, that placer gold may find lodgment in inequalities of the bedrock surface where no considerable deposition of detrital matter has taken place, though extensive placers are, as a rule, not formed because of irregularities in the bed-rock surface alone. The concentration of gold in river bars is analogous to its deposition in stream beds, for it is dropped where the velocity of the current is checked by the formation of eddies, due to the inequalities of the river floor.

A further study of this subject is made herein in connection with the more detailed analysis of stream action.

When the bed of a stream is the actual rock floor of the valley, it is called 'bedrock' in the true sense of the word. Later in its history the stream may flow on an aggraded bed of gravel or other sediment. If the stream gravels become covered with volcanic or other materials, the stream is obliged to flow over this new cover, called 'false bedrock.' Gold particles are normally deposited on or near the bed of the stream, which is called 'bedrock' or 'false bedrock' according to whether the bed is the true hard rock floor of the valley or whether it is a superficial layer of clay, volcanic tuff, or some such impervious material overlying previously deposited gravel. It is readily surmised, therefore, that there may be two or more tiers of gold-bearing stream channels, but the upper ones do not necessarily lie directly above the older and lower channels, and may not follow the direction of their courses at all. If the stream cuts clear down to the true bedrock, remnants of the older channels will lie at relatively higher positions instead of at lower horizons. In some cases, where gravel is deposited

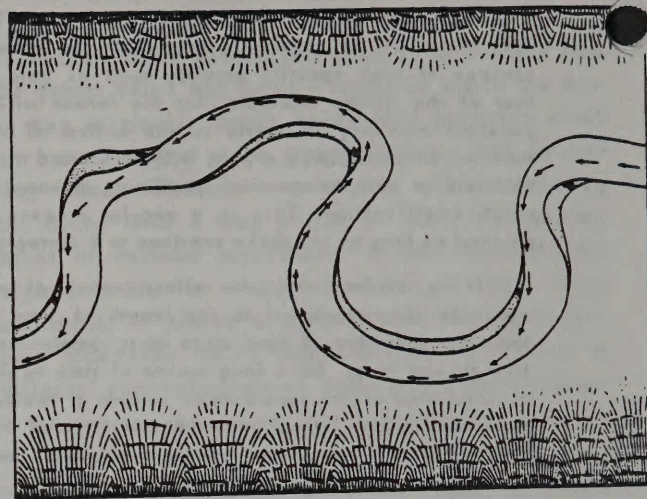


Diagram to illustrate the course of a river, indicating where gold particles are most likely to become concentrated. *(After Spurr, U.S.G.S., 18th An. Rept. 1898.)*

deeply on bedrock, forming a new aggraded bed, rejuvenation of the stream will stir up the entire mass of gravels, including the gold-bearing layer deposited on top, and the final result will be that most of th gold particles will reach a position very near bedrock. The complexity of the history of these processes is apparent; so also are the difficulties of the engineer who attempts to do a fair job of sampling.

In excavating for the Boulder Dam a sawed plank of lumber was found 60 feet deep, "lying under gravel on the edge of the inner gorge, a place that it could not have reached in any imaginable way except by burial during some comparatively recent flood." This case and many others show that the depth of burial by recent rivers does not necessarily mean that a great period of time has elapsed for the accumulation of the deposit. During high water, the whole mass may be stirred up and even boulders floated in the soapy mixture of heavy rock debris and water. This action gives the gold particles a chance to work their way toward the bottom of the mass.

For thousands of years particles of gold of various sizes, from nuggets to flour gold, were dropped and lodged in the riffles of bedrock along the natural river-sluices of the Sierra Nevada. Flattened particles are most easily carried: sometimes, suspended by air-films, tiny scales even float on surface of the water. Extremely fine grains of gold were swept by torrents through the canyons and out

into the Great Valley. In the present dredging grounds where they are found, they have been easily caught in false bedrock, which consists of clayey layers of volcanic tuff.

The very high specific gravity of gold, six or seven times that of quartz, with the ratio increasing to nine times under water, is the primary factor which causes this heavy resistant metal eventually to work its way to a point where it may sink no farther. Once it is caught on bedrock, the stream has great difficulty in picking it up again.

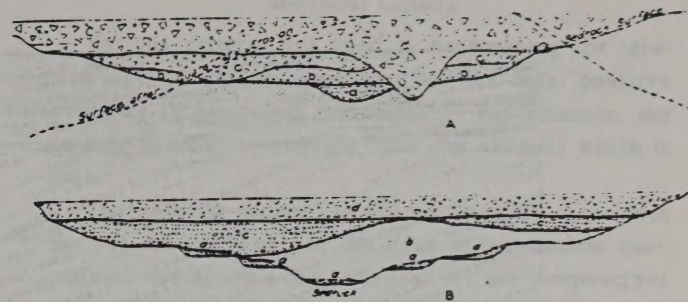
When a stream leaves its mountain canyon and enters a more level country or a still body of water, the materials carried by that stream are deposited in the form of a fan or delta. At the apex of this fan or delta fine gold may be deposited, and may never reach bedrock. It may be deposited on top of clayey, 'false bedrock' layers. The stirring action found to occur in rugged mountainous canyons during times of floods, which permits gold to reach bedrock, does not take place in the delta.

> By an odd paradox [stated Dr. Lindgren], gold is at the same time the easiest and most difficult mineral to recover. It is divisible to a high degree and owing to its insolubility the finest particles are preserved. A piece of gold worth one cent is without trouble divisible into 2000 parts, and one of these minute particles can readily be recognized in a pan.

Although gold is very malleable, and may be hammered into different shapes by stones hitting it as they tumble along in the stream, different particles are not welded together to form larger nuggets, as some people are prone to believe. Lindgren has shown that the largest masses of gold have come from lodes, not placers. Particles of gold may be broken down, however, from another piece. The more rounded or flattened nuggets have probably gone through more knocking about than the rougher pieces. These showing the original crystalline forms have probably not traveled far in the 'free' state.

It is also found that the more ancient placers, and those which have undergone many reconcentrations, contain gold of a higher degree of fineness than those whose source is near by, or in which the gold has not been deposited for such a long period of time. This may be due to the removal of alloyed silver by the dissolving action of surface waters.

The solution and reprecipitation of gold in the gravels is shown to be exceedingly rare or nonexistent, commercially. On the other hand, in some of the Tertiary channels, thin crusts of pyrite or marcasite are found deposited on the surface of the gold particles themselves.



A. Diagrammatic cross-section showing the four principal epochs of Tertiary gravel deposition in the Sierra Nevada. The deep gravels *a* represent Eocene; *b* to *d* are successively younger and probably represent Miocene stages for the most part. The rhyolite period is represented by *c* and the andesite by *d*.

B. Diagram showing deposits in the Deep Blue lead, Placerville. The older channel and benches of the inter-rhyolitic epoch are represented by *a*; rhyolite tuff, *b*; andesite cobble, *c*; andesite tuff-breccia, *d*. *Lindgren, U.S.G.S., P.P. 73.)*

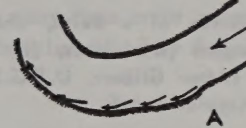


Diagram showing the place of greatest erosion on the bend of a river.

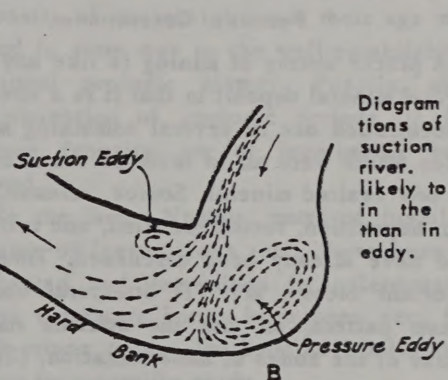


Diagram to show positions of pressure and suction eddies in a river. Gold is more likely to be deposited in the suction eddy than in the pressure eddy.

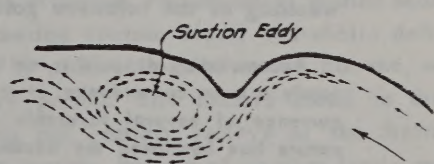
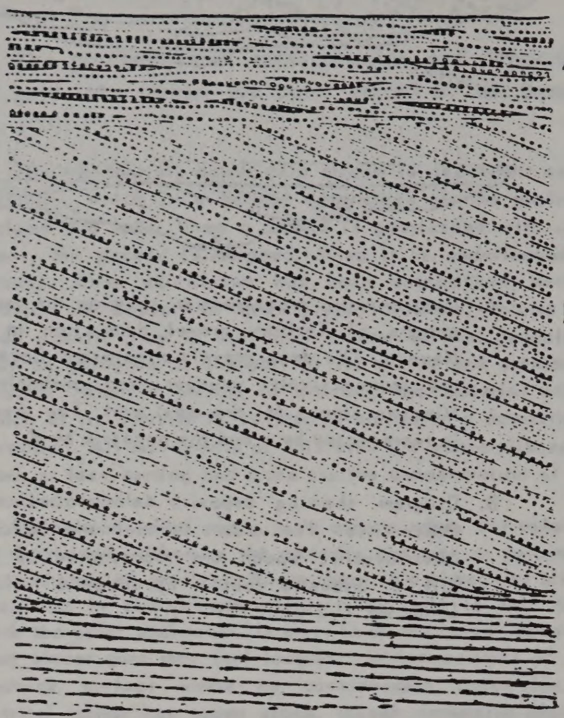


Diagram to show how a suction eddy is formed in a river. (After Thomas and Watt, see Ries and Watson, Eng. Geol.)



Ideal vertical section of a delta, showing the typical succession of strata: (A) topset, (B) foreset, and (C) bottomset, beds. *(After Gilbert, U.S.G.S., 5th An. Rept., 1883.)*

Factors of Concentration

A placer worthy of mining is like any other commercial mineral deposit in that it is a special case of concentration due to several combining natural processes which were all in favor of the accumulation of the one desired mineral. Source, release, transportation, deposition, reconcentration, and retention of the gold have already been discussed. Three extremely important factors are: (1) Structural control of the stream pattern, so that the streams run along the course of the zones of mineralization; (2) Decay and disintegration at the surface of the mineralized rocks prior to erosion; (3) A change in the cycle of erosion, causing rejuvenated flow of streams and the rapid washing of the released gold into stream channels.

Accumulation of gold in an important placer deposit is rarely a mere coincidence; it is rather the fortuitous concurrence of several favorable factors. In regions where nature has bestowed the advantages of extensive mineralization, rapid rock decay, and well-developed stream patterns, a relatively large amount of gold placer may be formed. But even in this ideal case, the favorableness of these several important factors must be assumed.

The general considerations which favor the accumulation of gold in special locations have been frequently discussed. Physically, the phenomenon is simple; in such locations where the gold has been deposited, the transporting power of the stream has become insufficient to carry away the particles of gold that have settled. The richness of the deposit will therefore depend not only upon the completeness of this loss of transporting power, and on the ability of the bedrock to hold the deposited gold at this point, but also, most importantly, on the general relationship of the gold sources to the stream. The early miners untiringly sought the 'ledge' or 'mother lode' which furnished certain rich placers. However, with the presence of mineralized zones as a source of the gold, the richness of a placer is perhaps due more to the efficiency of the stream as a concentrating device than to its uncovering rich lode deposits.

The ability of a stream to transport materials is essentially dependent upon the velocity of the water and the area and specific gravity of the particles of material being carried. The transporting power of water varies approximately as the sixth power of the velocity. This means that even small velocity changes have an enormous effect upon transporting power, ranging rather abruptly from velocities which can not transport an appreciable amount of gold to those which easily transport much of the gold that may enter the stream or be released from the gravels therein. The velocity of water is a complex relationship of grade, shape, and size of the channel, quantity of water, and other factors. A grade ranging from 30 to approximately 100 ft. per mile will favor the deposition of gold. With appropriate conditions of flow, these limits may be somewhat increased or reduced without serious handicap. When considering the grades of the ancient channels, however, one must remember that faulting and regional tilt often have considerably modified the original grade.

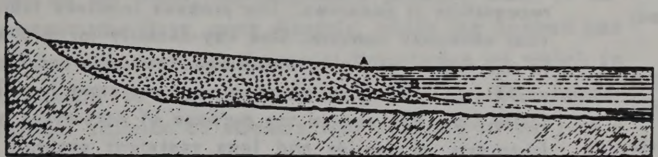
For the richest accumulations, the erosional conditions must be well balanced, so as to provide a long period of concentration. Slight uplifts tend to rework and further enrich placer deposits, as do increased volumes of water, inasmuch as both of these factors tend to increase the velocity. Local variations in the shape of the channels are of most interest, however, because they are immediately responsible for specific deposits of placer gold. When a stream canyon widens out, deepens, turns, or joins another watercourse, certain zones of concentration will be formed where the water velocities have been somewhat reduced and where eddy currents occur. These reductions in velocity immediately allow gold and heavy mineral particles to separate from the mass of gravel that is being carried and rolled down the canyon. Gold has a specific gravity of approximately six times that of the gravel, but under water this ratio becomes about nine times. This large gravity difference permits the gold quickly to work its way to bedrock and into any crevices therein. Here it remains, requiring excessive erosion to remove it to new locations.

In order that a major deposition of gold may occur, there must be an abundance of source material which contains more or less gold and which may be more or less easily eroded. A decayed formation of low-grade material could easily furnish more gold than a hard, higher-grade deposit. The decomposed material also supplies more gravel yield. balanced conditions of stream transportation, providing

that overloading or choking is minimized by uplifts or increasing water volumes. Plainly, a stream running along a vein system will have a greater opportunity to accumulate gold than one merely crossing it. Bedrock-controlled streams, therefore, provide a maximum contact with source material.

A further and very important factor is the ability of bedrock to hold the deposited gold in spite of the scouring action of the stream at higher water stages. A smooth, hard bedrock is a very poor one for placer accumulations. Bedrock formations which are decomposed or possess cracks and crevices are good, and those of a clayey or of a schistose nature are excellent in their ability to retain particles of gold.

Gold tends to resist most stream transportation. Coarser gold will migrate downstream an amazingly short distance from its apparent source throughout a long erosion period. The fine gold, which the stream can transport, is dropped rather completely within a restricted area at the mouth of the stream canyon.



Ideal cross-section of a delta, showing (A) topset, (B) foreset, and (C) bottomset, beds. *(After Gilbert, U.S.G.S. 5th An. Rept. 1883.)*

AGE OF PLACERS

Significance

The geologic age of a placer deposit is often a factor of primary economic interest. In the Sierra Nevada, the oldest system of Tertiary channels has proved to be the richest, for it was formed prior to the extensive volcanic activity which resulted in the covering of the mineralized bedrock surfaces as well as the valleys in which gold-bearing streams flowed. These streams, which cut directly through the mineralized zones, had an ideal opportunity to tap the primary gold resources of the region, while those which flowed only over a barren volcanic cover remained themselves barren of gold. In the region of the buried channels of the Sierra Nevada, many stream deposits of different periods are now found intermingled. To decipher their history and relative age is an essential part of the exploration geologist's work, in his search for the channels of greatest possible economic consequence.

Structural Criteria

Various criteria are used in determining the relative age of stream deposits, but the most positive evidence is structural relationship. For example, the cutting channel is younger than the channel which it cuts.

The deepest channel is not necessarily the oldest nor yet the youngest. In the case of the modern canyons of the Sierra — the youngest are the deepest; yet along the western foot of the range the present streams flow over older channels buried beneath. Where a canyon is filled with detritus or with lava, the newer stream flows on top of the deposit and is therefore higher than the old stream-bed beneath, while still more ancient stream terraces or benches may lie at higher and varying elevations on either side of the canyon. Some benches, representing former streams, may have even been left prior to a lava flow covering the deepest channel, while other benches may have been left later.

It is apparent, therefore, that the subjects of historical sequence and relative age are matters of detailed geologic and physiographic study which deserve more than mere superficial examination. They cannot be classified by dogmatic rules.

Paleontologic Criteria

In order to assign definite geologic periods to the deposits of ancient streams, their age should be related in some way to the well-established epochs of regional geologic history. Fossils, diagnostic in determination of geologic periods, if found in the stream deposits, are of inestimable value in this regard.

In the Sierra Nevada, parts of fossil plants consisting of leaves, logs, etc., have been extensively collected and determined by paleobotanists. Also, some vertebrate bones have been sent from the old drift mines to the Smithsonian Institution and elsewhere for scientific study.

Geologic periods the world over have been established largely on their marine fauna rather than their continental flora or fauna. For instance, marine beds of the Ione formation contain fossil sea-shells definitely assigned by paleontologists to the Eocene, or earliest Tertiary period. The fossils found in the sediments filling the Tertiary valleys of the Sierra are not, however, marine, but are of ancient lands and lakes. Correlation of geologic age by means of these land plants and land animals brings in complications that have not yet permitted the two bases of criteria,

marine and nonmarine, to be perfectly coordinated. Besides, most of the fossil leaves occur in tuffaceous lake beds that overlie the gold-bearing quartz-gravel deposits, and therefore do not give much direct evidence as to the age of the latter. Fossil wood, so common in the most ancient of the gold-bearing gravels is not as yet determinable nor diagnostic. In most cases it represents unstudied tropical forms which have been placed in the Eocene by paleobotanists because of the known climate of that period. The older gravel containing this fossil wood has previously been referred to the Cretaceous. In the Smartsville quadrangle, for example, a deposit containing petrified logs of probable Eocene age is described by Lindgren as follows:

The high, isolated area of well-washed gravel 3 miles north-northwest of Montezuma Hill is noteworthy; it is so much higher than the adjacent gravel channel of North San Juan that it must be assumed to belong to an earlier period; very likely it is of Cretaceous age.

It is a fact, however, that no fossils indicating a Cretaceous age have yet been found in these older gravels. Wherever definite marine Cretaceous beds do occur on the western foot of the Sierra Nevada the oldest stream channels of the vicinity are found to cut the Mesozoic sediments, showing a profound difference in age between the two.

R.W. Chaney has summarized the results of paleobotanical study of the fossil plants found in the Sierra Nevada as follows:

The tuffs and shales in which fossil plants occur interbedded in the Auriferous Gravels range in age from lowermost Eocene to upper Miocene. During this time there was a climatic trend from subtropical to temperate conditions, which resulted in the elimination of palms and other large-leafed species and the incoming of types of plants similar, in general, to those now living in North America. The Miocene flora indicating a temperate climate, includes genera no longer living in western America, although they occur in eastern America and eastern Asia. The evidences of difference in living conditions in the Eocene and the Miocene make it possible readily to differentiate between the older and the younger floras of the Auriferous Gravels.

Most of the fossil vertebrate bones described from the drift mines of the Sierra Nevada have been collected and sent in to paleontologists by persons who did not record their definite location, so that the exact geologic formations in which the fossils were embedded are unknown.

The paleontologist who works with vertebrate remains is not always apt to apply the same age to beds as that which has been assigned them by the paleobotanist; generally the former assigns a younger age. Many of the well-established Sierran Tertiary as well as later beds containing vertebrate bones were once given the blanket designation of Pleistocene.

Physiographic Criteria

Age correlation has sometimes been done purely on physiographic evidence. F.E. Matthes assigns Eocene, Miocene, Pliocene, and Quaternary times of uplifts to the various old surfaces found in the Yosemite region, tying in the Miocene surface correlation with geologic features of a fossil leaf locality occurring in the Tuolumne Table Mountain region. The fact that old surfaces have been resurrected during the Pleistocene has only recently [1934] been given consideration.

Quaternary erosion resulting in the uncovering of Tertiary volcanic ash and the resurrection of early Tertiary surfaces, formerly cut into pre-Cretaceous bedrock along the western flank of the Sierra Nevada, is a widespread geologic process which has heretofore not received the recognition it deserves. The process involves features of vast economic concern. One key locality for this study is in the region of Table Mountain, Tuolumne and Calaveras counties, California, where a very resistant late Tertiary latite flow has served the purpose of preserving not only fragments of earlier and less resistant geologic bodies consisting of volcanic materials, mud-flows, lake beds, and stream gravels of different ages, but also the underlying bed-rock surfaces of earlier Tertiary age. The ancient surfaces, the topography of which appears to have been controlled by bedrock structure, may be found in various stages of resurrection. In this area, the gold-bearing gravels were mined in ancient channels that ran in directions opposite or at an angle to the Table Mountain Channel; the latter apparently never contained any appreciable amount of gold-bearing gravel, contrary to the common belief. Though unmantled and dissected through Pleistocene and Recent epochs fragments of upland peneplained bedrock and in places gravel-covered surfaces are actually early Tertiary land forms, which have been brought to light after having been buried throughout later Tertiary volcanic epochs

Lithologic Criteria

The nature and composition of the material filling the ancient channels and valleys also indicate to what geologic period a deposit may belong. Gold-bearing gravels, composed purely of sand and quartz-pebbles, or of the bedrock complex, indicate that the channel is of the pre-volcanic period possibly Eocene in age. Most of the rhyolites were apparently formed during the latter part of this period and in the Oligocene. The most abundant of the volcanic rocks are composed of andesites, which seem to be largely of late Miocene or early Pliocene age. During the late Pliocene and early Pleistocene there were major basalt flows. Tuolumne Table Mountain is composed of latite, probably of late Pliocene age.

In the Recent epoch much pumice has been expelled from craters and blown over various parts of the Sierra Nevada.

Streams may always be considered younger than the rocks from which their gravel has been derived, though mudflows may receive their materials from active volcanoes. There is here an opportunity for a petrographic study of both volcanic rocks and the materials of the sediments. Special detailed analysis of stream correlation might be performed by means of the method known as "heavy mineral separation," previously mentioned as widely employed in petroleum geology.

LIFE HISTORY AND HABIT OF STREAMS

Need for Scientific Background

Exploration of placers and various ancient stream channels requires an understanding of the habits and life history of streams in general. This includes, on the one hand, processes of erosion and deposition, and on the other, physiographic history. Each is important; the first, more directly, while the second has to do with regional features, knowledge of which is essential to the exploration geologist. The fundamental science of streams was outlined in 1877 in a simple yet splendid manner by G.K. Gilbert in his masterpiece on the Henry Mountains, Utah.

Such natural processes as those related to streams are so universal that a study of them in one part of the world may be applied to conditions found in another. Similarly, an understanding of the ancient Tertiary streams of the Sierra may be gained by applying the knowledge of processes found in operation today where conditions and environments would appear similar.

In a province, such as the Sierra Nevada, where the development of the drainage system has been repeatedly interrupted by earth movements or by burial as a result of volcanic mud washes and lava flows, the history of the stream of any one chronological horizon is a separate entity, and may be entirely different in form and pattern from either earlier or subsequent systems. This fact, together with the complexity of any one system presents a problem much involved.

The need for a scientific background in the study of streams is therefore apparent. The more pertinent features of this study, together with its terminology, are outlined here.

Stream Erosion

Stream or *fluvial* erosion is complex. It may be divided into the several processes, *hydraulic action, abrasion, solution, and transportation.* A brief statement of the essential factors which control erosion is quoted from an authoritative textbook * as follows:

The capacity of a stream to erode, depends on its volume and velocity. The velocity in turn depends on (1) the slope down which the stream is flowing, (2) volume of water, (3) the shape of its channel, and (4) weight and volume of its load.

The rate of descent of the bed of a stream is the *stream gradient.* It is ordinarily expressed as so many feet per mile. The gradient changes from place to place along the course of the stream. Velocity increases rapidly with increase of gradient. Thus mountain streams with high gradients erode their valleys much more quickly than lowland streams of comparable size with low gradients. It follows that streams wear high gradients down to low ones by continued erosion, and that as the gradients are worn down the rate of erosion must decrease.

The volume of water is a variable factor in all streams, largely because of fluctuations in rainfall. Velacity and rate of erosion in any stream are therefore always changing. As a rule, these changes are too slight to be readily noticeable, but in some regions they are great enough to cause streams to dry up at certain seasons, and to rise in floods at others. In other regions, fluctuations are less extreme. Every spring the lower Mississippi has a normal rise in water level of 15 to 20 feet. The Nile normally rises 24 feet and the Ganges 32 feet. The erosive effect of such floods is considered below.

The shape of the stream channel as seen in cross-section also influences velocity. Since friction between water and channel slows a stream down, velocity is greatest in channels with the smallest area in proportion to volume of water. Deep, narrow channels therefore give greater steam velocity than broad, shallow ones.

A stream continues to acquire a load until it is carrying the greatest possible amount permitted by the gradient, volume of water, and kind of material available.

The "laws of erosive power" concern both transportation and abrasive power of the stream. If all the fragments of rocks had the same specific gravity, then the following definite action would take place.

If the velocity of a stream be doubled, the diameters of rock fragments it can move are increased 4 times. In other words, *the maximum diameter of the individual rock fragments a stream can move varies as the square of the velocity*... Calculations have shown that doubling the velocity of a stream increases its abrasive power at least 4 times, and under certain conditions as much as 64 times. In other words, *abrasive power varies between the square and the sixth power of the velocity.*

These laws not only explain the vastly greater erosion accomplished by swift streams than by slow ones under normal conditions, but they show clearly why exceptional floods; greatly increasing velocity by increasing volume, have such tremendous destructive power. The volume of

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*Longwell, Knopf, and Flint, A Textbook of Geology, Part I, Physical Geology: John Wiley & Sons, 1932, pp. 42-44.

the Colorado River measured at Yuma, Arizona, during a flood in 1921, was 155 times its normal volume. Again, when the St. Francis dam near Los Angeles gave way in 1928 and flooded the valley below, huge blocks of concrete weighing up to 10,000 tons each, were moved by the escaping water. In India, during the Gohna flood of 1895, which lasted just four hours, the water picked up and transported such quantities of gravel that through the first 13 miles of its course the stream made a continuous gravel deposit from 50 to 234 feet thick.

Preparation of Material Removed by Erosion

As previously described, the materials which are removed and washed into the streams are first prepared through *weathering* processes. Particles are loosened from the outcrop by surface disintegration, consisting largely of oxidation, hydration, and solution.

Since climatic environments were different in the past than they are now, the subject of ancient climates is an important problem in its relation to the development of ancient stream channels. The study of fossils imbedded in the deposits gives the most important clue to the nature of ancient climates. The condition and composition of the sediments themselves give another.

Transportation

The subject of river engineering brought forth at an early date much definite information as regards the carrying power of streams. The following statement is quoted from David Stevenson: *(Principles... of canal and river engineering*, p. 361).

The following are results deducted from experiments... on the size of detrital particles which streams flowing with different velocities are said to be capable of carrying:

3 in. per sec.—0.170 mile per hour, will just begin to work on fine clay.
6 in. per sec.—0.340 mile per hour, will lift fine sand.
8 in. per sec.—0.4545 mile per hour, will lift sand as coarse as linseed.
10 in. per sec.—0.5 mile per hour, will lift gravel the size of peas.
12 in. per sec.—0.6819 mile per hour, will sweep along gravel the size of beans.
24 in. per sec.—1.3638 miles per hour, will roll along rounded pebbles 1 inch in diameter.
3 ft. per sec.—2.045 miles per hour, will sweep along slippery angular stones the size of a hen's egg.

The following table is quoted from F.C. Gilbert (*Engineering Journal*, 1932, p. 4) maximum diameters of boulders which can be moved in sluices at certain velocities:

Diameter (inches)	Velocity (feet per second)
2	3.3
4	5.3
6	6.2
8	7.4
10	8.4
12	9.1
16	10.8
20	11.9
24	13.0
30	13.7

Transportation is done by the carrying of materials in *solution*, through *suspension*, and by the process of *saltation*. The materials thus carried are deposited by *precipitation* from solution, *sedimentation* from suspension, and *grounding* after 'leaping' along by that process called 'saltation.'

It has been shown by G.K. Gilbert, who carried on extensive laboratory experiments with running water, that the materials borne in suspension are easily enough sampled and their quantity measured; but the "bed load" is much less accessible. The load is carried forward by sliding or rolling along a smooth channel bed, as well as by saltation, which takes place when the bed is uneven and causes the particle to move irregularly in a series of jumps.

Gilbert calls the transportation of the bed load *hydraulic traction* in contrast to *hydraulic suspension*. His summary of "Modes of transportation, collective movement" is expressed as follows:

When the conditions are such that the bed load is small, the bed is molded into hills called dunes, which travel downstream. Their mode of advance is like that of eolian dunes, the current eroding their upstream faces and depositing the eroded material on the downstream faces. With any progressive change of conditions tending to increase the load, the dunes eventually disappear and the debris surface becomes smooth. The smooth phase is in turn succeeded by a second rhythmic phase, in which a system of hills travels upstream. These are called antidunes; and their movement is accomplished by erosion on the downstream face. Both rhythms of debris movement are initiated by rhythms of water movement.

In showing how complicated a stream's action may be, Gilbert states:

The flow of a stream is a complex process, involving interactions which have thus far baffled mechanical analysis. Stream traction is not only a function of stream flow

but itself adds a complication. Some realization of the complexity may be achieved by considering briefly certain of the conditions which modify the capacity of a stream to transport debris along its bed. Width is a factor; a broad channel carries more than a narrow one. Velocity is a factor; the quantity of debris carried varies greatly for small changes in the velocity along the bed. Bed velocity is affected by slope and also by depth, increasing with each factor; and depth is affected by discharge and also by slope. If there is diversity of velocity from place to place over the bed, more debris is carried than if the average velocity everywhere prevails, and the greater the diversity the greater the carrying power of the stream. Size of transported particles is a factor, a greater weight of fine debris being carried than of coarse. The density of debris is a factor, a low specific gravity being favorable. The shapes of particles affect traction, but the nature of this influence is not well understood. An important factor is found in form of channel, efficiency being affected by turns and curvature and also by the relation of depth to width. The friction between current and banks is a factor and therefore likewise the nature of the banks. So, too, is the viscosity of the water, a property varying with temperature and also with impurities, whether dissolved or suspended.

Gilbert classifies streams according to their transportational characters:

The classification of streams here given has no other purpose than to afford a terminology convenient to the subject of debris transportation.

When the debris supplied to a stream is less than its capacity the stream erodes its bed, and if the condition is other than temporary the current reaches bedrock. The dragging of the load over the rock wears, or abrades, or corrades it. When the supply of debris equals or exceeds the capacity of the stream bedrock is not reached by the current, but the stream bed is constituted wholly of debris. Some streams with beds of debris have channel walls of rock, which rigidly limit their width and otherwise restrain their development. Most streams with beds of debris have one or both banks of previously deposited debris or alluvium, and these streams are able to shift courses by eroding their banks. The several conditions thus outlined will be indicated by speaking of streams as *corrading*, or *rock-walled*, or *alluvial*. In strictness, these terms apply to local phases of stream habit rather than to entire streams. Most rivers and many creeks are corrading streams in parts of their courses and alluvial in other parts.

Whenever and wherever a stream's capacity is overtaxed by the supply of debris brought from points above a deposit is made, building up the bed. If the supply is less than the capacity, and if the bed is of debris, erosion results. Through these processes streams adjust their profiles to their supplies of debris. The process of adjustment is called gradation; a stream which builds up its bed is said to aggrade and one which reduces it is said to degrade.

An alluvial stream is usually an aggrading stream also; and when that is the case it is bordered by an alluvial plain called a flood plain, over which the water spreads in time of flood.

If the general slope descended by an alluvial stream is relatively steep, its course is relatively direct and the bends to right and left are of small angular amount. If the general slope is relatively gentle, the stream winds in an intricate manner; part of its course may be in directions opposite to the general course, and some of its curves may swing through 180° or more. This distinction is embodied in the terms *direct alluvial* stream and *meandering* stream. The particular magnitude of general slope by which the two classes are separated is greater for small streams than for large. Because fineness is one of the conditions determining the general slope of an alluvial plain, and because the gentler slopes go with the finer alluvium, it is true in the main that meandering streams are associated with fine alluvium.

Commenting on the curvature of a channel, which greatly complicates the transportation and deposition of debris by a stream, Gilbert says:

In a straight channel the current is swifter near the middle than near the sides and is swifter above mid-depth than below. On arriving at a bend the whole stream resists change of course, but the resistance is more effective for the swifter parts of the stream than for the slower. The upper central part is deflected least and projects itself against the outer bank. In so doing it displaces the slow-flowing water previously near the bank, and that water descends obliquely. The descending water displaces in turn the slow-flowing lower water, which is crowded toward the inner bank, while the water previously near that bank moves toward the middle as an upper layer. One general result is a twisting movement, the upper parts of the current tending toward the outer bank and the lower toward the inner. Another result is that the swiftest current is no longer medial, but is near the outer or concave bank. Connected with these two is a gradation of velocities across the bottom, the greater velocities being near the outer bank. The bed velocities near the outer bank are not only much greater than those near the inner bank but they are greater than any bed velocities in a relatively straight part of the stream. They have therefore greater capacity for traction, and by increasing the tractional load they erode until an equilibrium is attained. On the other hand, the currents which, crossing the bed obliquely, approach the inner bend are slackening currents, and they deposit what they can no longer carry.

It results that the cross section on a curve is asymmetric, the greatest depth being near the outer bank. As the winding stream changes the direction of its curvature from one side to the other, the twisting system of current filaments is reversed, and with it the system of depth, but the process of change includes a phase with more equable distribution of velocities, and this phase produces a shoal separating the two deeps. The shoal does not cross the channel in a direction at right angles to its sides but is somewhat oblique in position, tending to run from the inner bank of one curve to the inner bank of the other. In meandering streams it is usually narrow and is appropriately called a bar. In direct alluvial streams, where bends are apt to be separated by long, nearly straight reaches, it is usually broad and may for a distance occupy the entire width of the channel.

Deposition

The nature of stream or fluvial deposits and their detailed structure and texture are described in various textbooks, but not with sufficient detail to explain all the types of complicated features found in gravel deposits, especially complex deposits such as those of a mountainous region like the Sierra Nevada.

The constructive process of fluvial deposition goes forward side by side with fluvial erosion. This is a result of the complexity and variability of the stream currents, which constantly drop some rock fragments to the bottom while they pick up others. When a stream is actively eroding its bed at a certain point, it is merely picking up and carrying away more rock material than it is depositing there, and when it is actively depositing the reverse is going on. Therefore whereas fluvial erosion and deposition are processes physically opposed to each other, they can be separated in practice only by recognizing the preponderance of one over the other.

The arrangement of materials deposited in a delta, however, is well known, and gives a picture which is more or less duplicated whenever the current of a stream is checked by a body of standing water and the materials transported are permitted to drop. The term *foreset beds* is applied to the deposition on the frontal slope of the growing embankment. *Bottomset beds* are of finer grain and are formed by the particles carried out beyond the slope and deposited in deeper water. *Topset beds* are composed of the materials laid down and spread out on top of the other materials by the fluctuating stream.

The material deposited by a stream is called alluvium and makes up *fan, floodplain,* and *delta* deposits. The term *fanglomerate* is used for the gravel materials of alluvial fans.

An alluvial fan is built up at the point of abrupt change in gradient of a loaded stream. A floodplain is a series of coalescing alluvial flats along a valley. A delta is the final deposit by a stream, unloaded as it enters a still body of water.

Overflow of a stream onto its floodplain will cause *natural levees* to be built up as low ridges bordering the channel. Lateral swinging of a stream causes cutting on the outer sides of the curves and deposition on the inside, which results in the widening of the valley. A meandering stream may develop to the point of straightening itself in places by the cutting off and silting up of the meanders, forming *oxbow* lakes as a result. A stream which forms a complex interlocking network on its floodplain typifies *anastomotic* drainage. An overloaded stream on a low gradient, becoming choked, and constantly obliged to cut new channels, develops an intricate network on a floodplain; the process is termed *braiding*.

If the stream is rejuvenated and therefore cuts deeper, floodplain *terraces* or *benches* are formed. The benches may, however, be cut and left in the bedrock and covered with only a film of gravel on the surface.

Streams composed entirely of thick mud, called *mudflows,* are akin to landslides.

Another normal though infrequently operative process in arid regions is the *mudflow*. It occurs only where fine rock material becomes water-soaked on steep slopes after heavy rains, and moves downward as a slippery mass.... It advances in waves, stopping when it becomes too viscous to flow and damming the water behind it until it liquefies and again proceeds like an advancing flow of lava. Mudflows can carry boulders many feet in diameter. Observers have seen these great rocks bobbing 'like corks in a surf.' Successive mudflows play a part in the building of fans. (Longwell, Knopf, and Flint, pp-78.)

The transporting power of mudflows, their various peculiarities, and the resultant unsorted deposits have many characteristics much like those of glacial deposits and have frequently deceived engineers.

The distinguishing characters of glacial deposits are clearly summarized by Eliot Blackwelder, who has made a special study of them:

The deposits left by glaciers should be distinguished from those made by streams, lakes and other agencies.

The ice tongue of a glacier leaves only one type of deposit called *till*. It is wholly unstratified and its components are quite unsorted—a jumbled orderless mass of clay, sand, and boulders. Blocks three to five feet in diameter are common and those 25 feet or more are not rare. In general, the size of such boulders depends upon the spacing of the joint-cracks in the rocks of the mountain sides. Usually till is an earthy mass well sprinkled with stones and boulders but in some cases the boulders predominate. This is particularly true of the deposits of small glaciers which have done little more than sweep the coarse talus from the valley slopes. The stones in till may be of any shape from well-rounded to angular but many have the corners and edges rounded. It is usual to find some that have been bevelled by being rasped along the bottom of the glacier. Hard rocks may thus be well polished. Such stones, like the bedrock, are covered with scratches which are easily recognized.

It is often difficult to identify till, especially if it has been much decayed or eroded or is poorly exposed. It may then be confused with other bouldery deposits which are unstratified. From volcanic mudflow deposits, such as abound in the Miocene beds on the Sierra Nevada flanks, till may often be distinguished by its containing large quantities of nonvolcanic rocks. Even this criterion fails where glaciers occupied volcanic mountains such as Mts. Shasta and Rainier. Ordinary mudflow deposits are seldom as thick as glacial moraines and are generally interbedded with typical stream gravel and sand, as in the alluvial fans of the arid regions. Unless the surface topography is still preserved or unless one finds plenty of scratched stones, it may be almost impossible to distinguish till

from landslide dumps. In many cases no one type of evidence can be relied on, but one must study all the facts and weigh the importance of each.

The rivers which issue from glaciers deposit coarse gravel, then fine gravel, and finally sand as the current weakens near the edge of the mountains. These three grades of detritus are more or less interbedded, because variations in the river's power occur from time to time at a given place. Like river deposits in general, those of glacial streams are distinctly stratified, though usually cross-bedded. They are fairly well sorted into separate layers of sand and gravel of various sizes. The pebbles are normally well rounded and very rarely either faceted or scratched. Angular stones are rare. Although small boulders are carried by ice cakes and become stranded in the glacial river gravel, large boulders are generally absent.

To distinguish the deposits of a glacial from a non-glacial river is difficult and often impossible, unless one can trace the gravel terraces into actual connection with a glacial moraine or can work out in detail the physiographic history of the district.

The deposits made in glacial lakes are rather distinctive. On the bottom of the lake, clay and silt are laid down very evenly in thin sheets which are commonly banded as seen later in cross-section. This is due to the fact that the layer deposited in winter is finer and darker in color than the one laid down during the summer melting season. Unlike most lake deposits the glacial lake clays commonly contain scattered pebbles and even small boulders which have been dropped from cakes of ice floating over the lake. These laminated clays may be associated with beds of peat or chalky diatomaceous earth formed by organisms that inhabited the clearer parts of the lake. Streams entering the lake form advancing deltas composed of gravel and sand in which the stratification is characteristic of deltas in general. In quantity the delta deposits often exceed the other lake deposits greatly, for the glacial rivers carry large quantities of coarse detritus all of which lodges in the deltas rather than upon the floor of the lake.

Other deposits that may be formed in glacial valleys, such as landslides, talus, and alluvial fans, need not be described specifically. They are local and generally well known.

The deepening of canyons and the deposition of gravels by outwash streams which issue from the snouts of glaciers form a very important chapter in the robbing and destruction of earlier gold-bearing gravels. Much of the material carried by Pleistocene glacial outwash streams of the Sierra Nevada was dumped at the foot of the western slope of the range at the point where the major rivers enter the Great Valley. The extensive gold-dredging ground of California to a large extent owes its existence to these streams.

So far as the processes involved in stream action are concerned, much can be gained by the detailed study of ancient glacial stream channels found throughout the world, especially in its northern belt.

Physiographic Terms Relating to Streams

The mere definition of some of the terms used in stream physiography gives a direct insight into the science.

Cycle of Erosion includes the series of changes from the *initial* cutting of a surface to the final reduction of a region to a *baselevel.* The surface of a region reduced to fairly low relief, but still undulating, is called a *peneplain* (also spelled *peneplane).* It is a significant fact that the Sierra region in early Tertiary time was approaching the peneplain stage of erosion, when the area was covered with lava to form a more nearly plain-like surface, and later uplifted. The uplift caused deep dissection by streams.

Stages of stream development, from gulleys to completely worndown plains, consist of *youth, maturity,* and *old age,* with continuous transitional stages between. The early stages represent very rapid growth, which slows down gradually until at old age the changes may be extremely slow.

The *genesis* or origin of the stream takes into consideration the initial surface over which the stream first flowed. Several terms are used by geologists in relation to this subject. *Consequent* streams are those whose positions were determined by the initial slopes of the land surface. *Subsequent* streams are those which are established by growing headward along belts of weak rocks.

Where the underlying structures of the rocks have affected the direction of the stream and its valley, the stream is said to have *structural control.* The terms *fault, joint, strike, anticlinal, synclinal,* and *monoclinal* are prefixed to the word 'valley'; thus, *fault-valley, strike-valley, etc.* It is especially significant that the richest gold-bearing channels have structural control – the streams originally ran on and along mineralized zones in bedrock.

Streams may start their courses over one sort of geological formation, but as time progresses they may cut through it and be *let down* on a lower and entirely different type of structure; such streams are said to be *superimposed* (or simply *superposed)* on the underlying rock structure. When a certain stream *pattern,* originally developed because of previous topographic or structural conditions, is retained even after those conditions are removed, the stream is said to have *inherited* its peculiar features from much earlier periods of its life.

Streams may be *intermittent* or *permanent.* Depression of topography along a coast may cause the sea to invade the valleys, and the streams are *drowned.* Uprise along the coast may leave *hanging valleys.* Tributaries to a main stream which has been much faster-cutting may also be left 'hanging'; as, for example in Yosemite Valley where the side streams reach the Merced River by way of beautiful waterfalls.

The *longitudinal profile* of a stream is taken from its source to its mouth, while its *gradient* represents

its inclination at some particular part of its course. Tributaries are said to be *accordant* when they enter at about the same level as their main stream. A stream is said to be *at grade* when rate of degradation and rate of aggradation are about equal.

A stream which is able to maintain its course, even when a segment of the earth is gradually raised athwart that course, is called an *antecedent* stream. If, however, the uprise of the mountain causes the flow of the streams to be accelerated down its slope so that they cut deeper gorges, they are said to be *rejuvenated.* Even stream *meanders,* developed on a plain, may be *entrenched* or *incised* deeply, to form a winding canyon by elevation of the plain.

Rejuvenation may be effected in other ways than mountain-making. Change of climate may make a decided change in stream cutting. Stream *piracy* is another important cause. This consists of the *capture* of

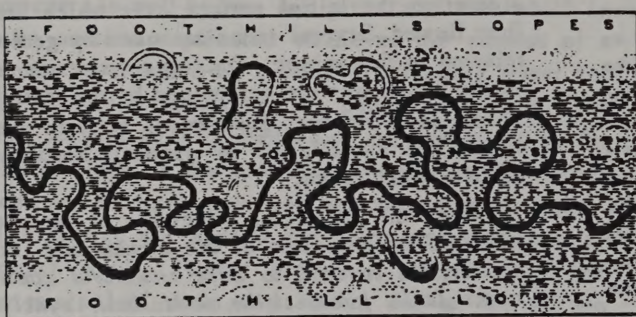


Diagram to show a meandering stream with oxbow loops. Such a stream develops in a valley worn down in base-level and not subject to extensive floods.

one stream by another. The second lies at a lower elevation; its head cuts back until the first is tapped, or *beheaded.* Then the water from the first stream, from that point upward, is caused to flow into the capturing stream. In this manner the flow in the first is accelerated often to such an extent that a new gorge may be formed. Whole stream systems may thus be *readjusted* and repeatedly go through new life cycles. Piracy and stream adjustment were apparently very active in the Sierra Nevada during Tertiary time; this process partly accounts for many of the deep accumulations there of Tertiary gravel.

The *pattern* of an individual stream, or of the whole or any part of its system, develops in its own peculiar way because of certain controlling geological, topographical, and climatic features. The pattern, therefore, is a character significant enough to bear special study and to support many new descriptive terms. It is now best studied by means of aerial photographs, though detailed topographic and geologic maps once presented the only bases of accurate expression.

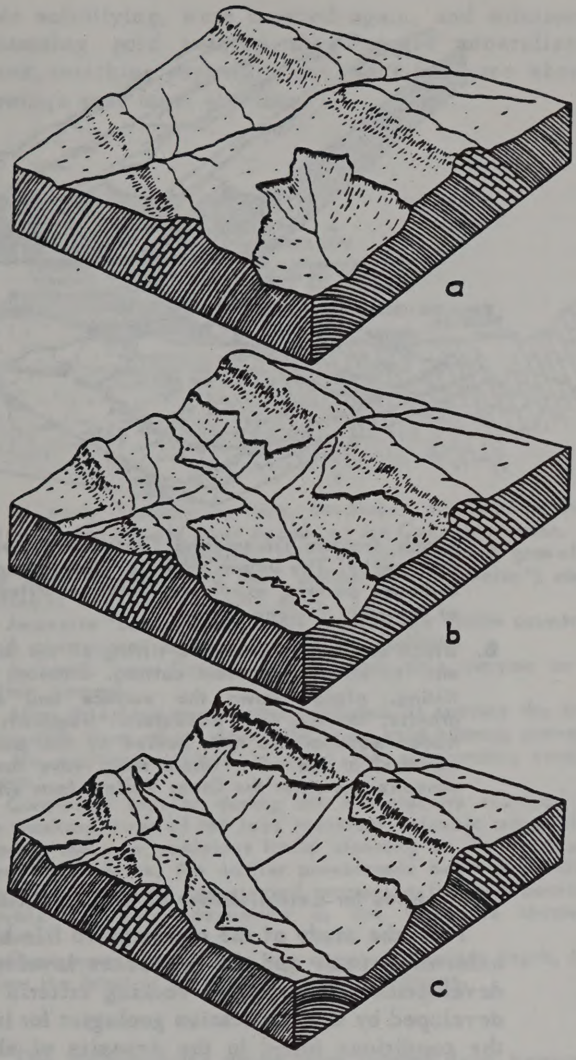
Many clues as to the geologic structure and history of the underlying region are gained by the study of stream pattern, from either an intensive or regional point of view. In the northern Sierra Nevada, the stream pattern developed prior to volcanic activity was structurally controlled by bedrock; but during the later period of volcanism it suffered change by that widespread activity. The major streams subsequent to volcanism followed the slope of the lava-covered, tilted, and uplifted fault-block range.

An instructive outline of the subject of stream pattern is given by Emile Zernitz who describes and illustrates by many actual examples such patterns as follows: *dendritic, trellis, rectangular, annular, radial,* and *parallel.* He states that:

The patterns which streams form are determined by inequalities of surface slope and inequalities of rock resistance. This being true, it is evident that drainage patterns may reflect original slope and original structure or may reflect the successive episodes by which the surface has been modified – including uplift, depression, tilting, warping, folding, faulting, and jointing, as well as deposition by the sea, glaciers, volcanoes, winds, and rivers. A single drainage pattern may be the result of several of these factors. *(Journal of Geology, vol. 10, p. 498).*

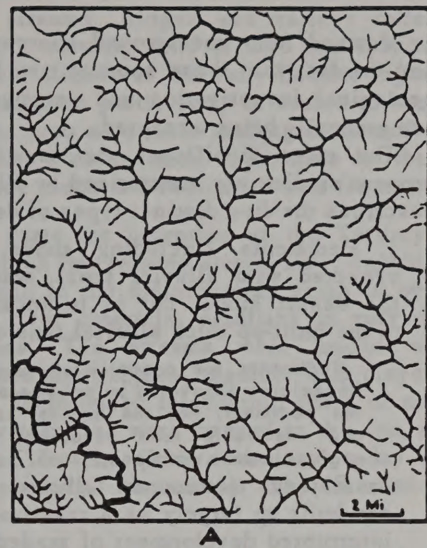
Diagram to show a subdividing or *anastomosing* stream in a valley subject to floods.

The fact that lakes fill depressions, basins, and valleys along stream courses, and that their deposits are intimately associated with those of streams, makes their study interrelated with that of stream channels. The so-called 'pipe-clay' deposits, which nearly always immediately overlie the gold-bearing gravels, represent beds of silt and finely divided volcanic ash, which have settled in lakes and formed a series of thin layers. They often contain impressions of leaves, showing the character of the forests that grew in that early period. This feature indicates that before the volcanic flows came to cover them the stream valleys had been transformed into lakes, into which the volcanic ash settled.



Diagrams to illustrate three successive stages in *stream piracy*. The stream cutting back at the lower elevation beheads and captures the stream flowing at a higher level.

In general, a lake is not as long-lived as a stream. Streams tend to destroy lakes, either by gradually filling their basins with detritus or by cutting down their outlets to a point where their basins may be drained. Sometimes, however, lakes persist long enough so that the area whose drainage they receive is worn down to a *local* and *temporary baselevel.* Lakes are formed in a number of ways: by landslides across stream courses; by lava flows which dam up the drainage: by the down-faulting of segments of the earth which are then filled with water; by glacial action, either by scooping out rock basins or by damming with till; by peculiar action of rivers themselves, such as silting off oxbow loops in a meandering course. An excellent paper, now out of print, was written by the late Dr. W.M. Davis. It not only discusses the present natural lakes of California, but the origin of lakes in general. *(California Journal of Mines and Geology, vol. 29, pp. 175-236).*



A. An example of *dendritic* drainage pattern, which develops where the underlying rocks lie in a horizontal position and offer uniform resistance to erosion.

B. An example of trellis drainage pattern, which develops in folded rocks differing in degrees of resistance to erosion. The stream courses are therefore *structurally controlled.*

Desert Processes

In the desert, the processes of erosion and of stream action differ very much from those in more humid areas. Only quite recently have the geologic processes in the desert been given much consideration; so also has much serious thought only lately turned toward the possible development of desert placers on a larger scale than mere 'dry washing'. The fact that adequate supplies of water may usually

be derived from underground sources in the desert, and the fact that these sources may be found through geological investigation and geophysical surveying, are gradually being accepted.

The most significant results of recent desert-process studies are summarized by Blackwelder as he describes the five distinct types of desert plains:

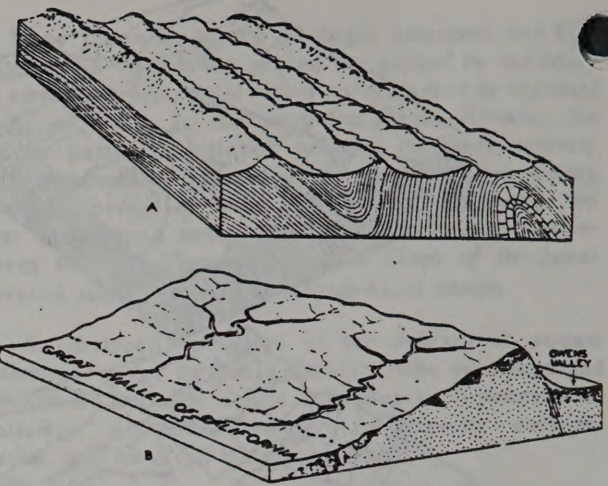
1. Pediments (including those only thinly veneered with alluvial fans), which represent the desert slope, cut in bedrock, in contrast to the built-up thick alluvial fans or bajadas.

> Pediments are essentially compound graded floodplains excavated by ephemeral streams ... the pediment, not the bajada, is the normal and inevitable form developed in the arid regions under stable conditions.

2. Bajadas (compound alluvial fans), which are built up largely as a result of disturbed or interrupted development of graded slopes. The upward movement of a fault block causes renewed erosional activity, and thick gravel deposits are formed by the consequent torrents and mudflows when they are released from their canyons and enter a region of lesser gradient.
3. Dried-up lake bottoms of the desert, or playas, whose conditions indicate that once more permanent lakes filled the flats and were fed by streams that are now nonexistent.
4. Dip slopes, which are broad planes developed on a denuded, hard, flat-lying or gently tilted rock layer.
5. River floodplains, which are abnormal desert features, but which were apparently more widespread at an earlier time, when precipitation in a given region was much greater than it is today.

Map of the southeastern margin of the San Joaquin Valley showing fans built by streams which disappear after leaving their mountain canyons. The coalescing alluvial fans form in this manner an extensive *bajada*.

These earlier plains and stream courses have been covered by the bajadas of today, so that the older channels have become buried in the true sense of the word. Some of them may represent a large potential placer reserve, but they have not yet been well investigated.



A. Block diagram illustrating Cretaceous Sierra Nevadan topography. The upturn edges of bedrock controlled the drainage pattern, which was later inherited by streams of the early Eocene period.

B. Block diagram to show the tilting of the Sierra Nevada and its effect on stream cutting. Erosion, prior to the tilting, planed down the surface and exposed the granite, leaving only occasional fragments of the intruded metamorphic rock bodies as *roof pendants*. The streams, at the point where they leave their mountain canyons and enter the Great Valley, form alluvial fans.

Need for Establishment of Working Criteria

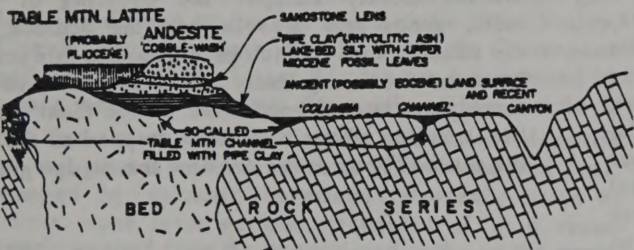
From the study of the complicated life history and habit of streams and the processes involved in their development, a series of working criteria should be developed by the exploration geologist for interpreting the conditions found in the deposits of all streams, including those of Tertiary age in the Sierra Nevada.

GEOLOGIC CONDITIONS IN THE GOLD PROVINCES OF CALIFORNIA

In the Sierra Nevada and Klamath Mountains of California gold-bearing quartz veins are generally found in metamorphic rocks not more than a few miles, at the most, from intrusive bodies of granitic rocks. The age of the metamorphics, which are made up of slates, schists, limestones, and meta-igneous bodies, is earlier than Cretaceous, namely pre-Paleozoic, Paleozoic, Triassic, and Jurassic. The time of intrusion of the granitic masses was late Jurassic.

The quartz veins were formed shortly after the igneous intrusion, during the last stages of the Jurassic. It would seem that the metamorphic rock masses were uplifted and intruded by molten magmas which then cooled and contracted, causing the surrounding and over-lying roof-rocks to crack along many planes of weakness and to form thousands of fissures. Into these openings the residual gaseous solutions, released from the crystallizing granites and composed largely of silica, were injected. These,

after solidifying, were crushed again, and solutions containing gold entered the complex mineralized zones, enriching especially the cross-fractures where openings were most abundant.



1. An early Tertiary surface, developed on bedrock over which flowed the old river system of the Columbia basin.
2. This surface, including its rich gold-bearing gravels, was later covered with lake sediments ('pipe-clay') consisting of fine silty volcanic ash.
3. Andesite 'cobble-wash' then covered the entire country.
4. A canyon was cut in this andesite mudflow.
5. A basic lava (latite) flowed down this canyon as a molten stream.
6. After this lava cooled and hardened, it became the bed of another river which deposited some gold-bearing gravels on its surface, washed in from the surrounding eroded region.
7. Continued erosion during the Pleistocene resulted in the washing away of the less resistant volcanic materials, leaving the very resistant latite standing as Table Mountain. Furthermore, the earlier prevolcanic bedrock surface was uncovered, or resurrected, exposing the gold-bearing gravels originally deposited in the Columbia channel system.
8. Quaternary erosion has cut canyons of great depth, far below the level of the former Tertiary surfaces.

The gold-bearing veins thus formed deep beneath the surface of the earth, had then to be brought to the light of day by the erosion and removal of the covering layer of rocks, nearly two miles in depth. This gigantic work was accomplished during the Cretaceous period, and as a result thousands of layers of shales, sandstones, and conglomerates several miles in stratigraphic thickness were laid down in an adjoining marine basin. Some of these beds—especially the last ones to be deposited—are gold-bearing, showing that the last part of the Cretaceous erosion finally reached the hidden veins.

That part of the geologic history, however, which was most important so far as the making of rich placer deposits is concerned, was the Eocene period. The deep erosion which took place during the Cretaceous had worn the surface down to such an extent that the metamorphic rocks with their mineralized zones had been reached, so that the streams of the Eocene ran along and through them.

Structural control of the drainage, fully developed during the Cretaceous, was thus inherited by the Eocene streams. Ridges and valleys followed the north-south trending beds of hard and soft rock. The subtropical climate of the early Eocene together with other conditions particularly favorable to rock disintegration, such as a more prolonged time of stability in the earth's crust, made it possible for the gold in the surface rocks to be released from its matrix.

Then came the inception of the Tertiary Sierran uplift, which rejuvenated stream flow, causing the released gold in the disintegrated veins to be washed into the river channels, resulting in very rich concentrations. The streams were loaded with fine quartz sand and pebbles, together with clays derived from the decomposed feldspathic parts of the rocks. The finer particles were washed to the sea, and as a result the Eocene (Ione formation) today contains large deposits of commercial clay interbedded with quartz sands.

The westward tilting and resulting acceleration of stream flow interrupted the north-south drainage system inherited from the Cretaceous period. The readjustment of the streams resulted in their general direction of flow being finally changed from north and south trends to a westerly course, somewhat as it is today. The longest of these known streams even headed far to the east into what is now Nevada.

Hardly had the Eocene come to a close when much rhyolite ash, thrown into the air from volcanoes, settled over the region. By Oligocene time, rhyolite ash had covered much of the northern Sierra Nevada, damming rivers and forming lakes, the bottom sediments of which are now represented by thinly layered pipe-clay immediately overlying the richer gold-bearing gravel. The newly developed rivers, flowing directly down the western-tilted slope of the Sierra, over a volcanic cover, as consequent drainage, were repeatedly interrupted by continued ejections of lava and a further tilting of the region. Not until the late Pliocene or early Pleistocene did the constant outpouring of lava cease. Then, by a series of violent earth movements, the Sierra Nevada broke away from the region to the east along huge fault-scarps, which are formed at the foot of the present steep eastern slope, where displacements are now measured in thousands of feet. Within the Sierran slope, smaller adjustment faults also broke the continuity of the older buried Tertiary stream grades. Some of the ancient streams, the courses of which headed farther to the east, were virtually 'chopped' into many pieces; some were elevated and others depressed, and many were warped to various peculiar positions. Undoubtedly there are some segments of these old channels which now lie deeply buried beneath great thickness of alluvium in down-dropped fault-blocks east of the Sierran escarpment.

In the Great Basin and the Mojave Desert region of California and Nevada are remnants of Tertiary stream deposits, interbedded with or lying beneath

lavas, all of which have suffered much by faulting and warping.

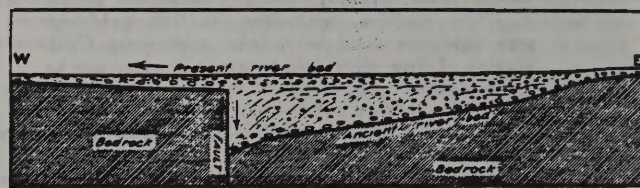
In these regions, however, the most important period of placer formation was in the early Pleistocene, rather than in the Tertiary. Two types of lode gold supplied the source. One type was formed in much the same manner as the Sierra Nevada lodes. The other consisted of mineralized zones in rhyolite of early and middle Tertiary time. In the Pleistocene there were normal streams flowing through the desert, fed by melting glaciers of the higher mountains. Placers that were formed by these Pleistocene streams have since been largely covered by desert alluvial fans. Some have been elevated and are cut by more recent streams, when present in this arid region, so that recent concentrations from older river gravels provide one source for the desert dry placer.

In the Klamath Mountains, though the early geologic history was much like that of the Sierra Nevada, there were no lavas to fill the valleys in which the stream gravels were deposited. Uplifts, accompanied by renewed stream-cutting, caused terraces to be left on the valley sides, where the rich gravels have given up their gold to hydraulic mining. Some of the finer gold particles were washed by the rivers to the sea and have formed deposits known as beach placers along the northern shore of California.

Down-faulting of various degrees of magnitude in places caused accumulations of gravels to form, especially on the down-throw sides of faults. A number of such faults are located in both the Sierra Nevada and Klamath Mountains, and may hold a reserve of gold not yet entirely recovered. In the Sierra most of these minor displacements show that the east side of the fault-plane has been dropped down, so that where faults cross westward flowing rivers, accumulations of gravels have taken place in pockets thus formed, east of the fault-plane and upstream.

The whole Pleistocene period was one of great events for California. The eastern side of the Sierra Nevada was raised to very lofty heights. The westward-flowing streams, as a consequence, were so greatly accelerated that they cut deep and rugged canyons. The uprise, accompanied by faulting, caused such violent earthquakes that enormous masses of rock were shaken from the mountain sides, in many places to form local lakes which were later to be drained and destroyed by active erosion. Glaciers developed in the higher mountains and crept down the canyons, carving them wider and leaving them U-shaped in form. Their melting supplied much water to the streams. Some local volcanic cones were built up here and there near or over the fault planes.

The Tertiary stream gravels, which had long been buried deeply beneath lavas, were exposed by the Pleistocene canyon-cutting rivers. From the dissec portions of the old channels, gold was removed washed into the newer streams, which concentrated it on their bedrock riffles. The remaining portions of the Tertiary deposits were left with their stubs exposed high up on the intervening ridges. In places, erosion merely stripped the covering of volcanic tuffs, sands and gravels from the bedrock, leaving the channel with its rich gold deposits laid practically bare for the lucky early miner to win. Some of the finer particles of gold were swept clear out to the Great Valley where they were dropped on the edge of the plain. These areas are now the dredge grounds.



Ideal cross-section of a river in the Sierra Nevada, the bed of which has suffered down-faulting on the upstream side, causing gravels, sand, and silt to accumulate in the pocket thus formed.

The general western tilt of the Sierra Nevada has been found to continue along the same slope (about two degrees) far beneath the alluvium and sediments of the Great Valley. Areas dredged for gold values in the gravels thrown down by the great canyon-cutting rivers of the range lie along the extreme western margin of the foothills near the place where bedrock passes beneath the alluvium, and aligned in a direction N.20° W. The gravels dredged do not lie directly on bedrock but on tuffaceous clay layers, spread out above the detritus-covered down-warped Sierran surface. Beneath the 'false-bedrock' and cut in the true bedrock surface is a stream pattern with gold-bearing gravel filled channels, now reached only in one or two places. This buried channel system undoubtedly holds in reserve a great wealth for future improved exploration and development. Excessive underground water is always encountered in these mines which are located beneath the level of the alluvial plain.

The great differences between the geology of the Coast Ranges and that of the Sierra and Klamath regions are fundamental in that the western area served frequently as a basin for deposition during the Tertiary and Cretaceous, while the latter represented land areas throughout that time. The Coast Ranges, together with the Great Valley now contain enormous accumulations of marine Tertiary and Cretaceous

sediments while the Sierra Nevada and Klamath Mountains are not so covered. Cretaceous and Tertiary streams coursing down the mountain flanks brought gravel, sands, and clays into a marginal sea.

The very fact that streams are conveyors of materials, in contrast to the basins of deposition toward which they flow, accounts for the very different geologic conditions on the two sides of the Great Valley. Certain geologic time divisions of the Cretaceous and Tertiary of the Coast Ranges are represented by strata measured in many thousands of feet, while in the Sierra mere films of Tertiary gravels, or deposits of no greater thickness than a few hundred feet, trapped by volcanic coverings, represent some of these same later periods. Particles of gold, recurrently washed from the mineralized rocks of the mountain range were dropped by reason of their high specific gravity, and retained in the bedrock riffles of both the ancient and modern streams, while the lighter detritus was carried to the broad sea basins to form strata covering hundreds of square miles.

CONCLUSION

The depletion of the more accessible and more easily discovered gold placers, followed by losses due to poorly directed exploration, calls for a more effective technique to bring further success to placer mining. The technique is available; the next thing to do is to apply it.

First, there is aerial photography which may speedily and accurately give a wealth of valuable information as regards geology, and in addition, the finest sort of a map showing surface features in greatest detail.

Second, there is geophysical surveying which, when coordinated with geology, may greatly aid underground prospecting in making new discoveries and in reducing its cost by more intelligently directing its course of action.

Third, physiographic geology, advanced to a more systematic science than ever before, may be used in unravelling the history of the ancient streams and their corresponding topography. Contouring the pre-lava surface is found to be an excellent method of showing graphically this ancient topography, and especially the old valleys in which lay the early gold-bearing streams.

Fourth, a better understanding of desert processes in general and desert placers in particular should help to develop a gold reserve which has so far not received the attention it deserves.

Fifth, the technique recently developed in the examination of stratified sediments, their structure, texture, mineral-grain composition, etc., may be aptly applied to placers, to aid in tracing out their origin and the courses of the older drainage systems now extinct.

In taking stock of the possible reserves of placer gold in California, several sources would seem worth investigating. All of these require detailed exploration prior to any attempt at mining. For the most part, these reserves are buried or concealed in such a way that they have either been overlooked or considered too remote or too much of a speculation for a mining venture. Such factors as involved water-rights, litigation, difficulty in gaining title, laws unfavorably affecting hydraulic mining, lack of sufficient capital, and many other stumbling blocks now prevent good placer ground from being worked.

The possible reserves discussed in this report may be summarized as follows:

Pleistocene and Recent Placers.

1. Deep river gravel deposits, over which the present larger rivers are now flowing. Recent and Pleistocene faulting caused gravel to be accumulated on the down-throw side of faults, while the rivers have continued to flow over the gravels without washing them completely out.
2. Pleistocene stream placers, buried beneath alluvial fans of the Great Basin and Mojave Desert.
3. Recent ephemeral stream deposits and alluvial fans or 'bajada placers' of the Great Basin and Mojave Desert regions.
4. Marine or beach placers along the coast, for the most part located in northern California.
5. Isolated high terraces or bench gravels, such as those which occur in the Klamath Mountains.

Tertiary Stream Placers

6. Gold-bearing channels cut in bedrock which lie beneath the false bedrock layers of the dredged areas along the western foot of the Sierra Nevada.
7. Buried Tertiary channels and associated covered benches located in the well-known gold-bearing districts of the state. Large areas still lie buried and unexplored in some of the older mining districts.
8. Buried Tertiary channels and benches in the lava-covered district which lies between the Sierra Nevada and Klamath Mountains. Most of this area is probably too, deeply covered to be reached by mining, but the southern marginal area may have some possibilities.
9. Tertiary channels of the Great Basin and Mojave Desert areas, interbedded with volcanic rocks or lying beneath them.
10. Tertiary marine placers. Finely divided gold particles in the Ione formation at the point where the corresponding Eocene streams entered the Ione sea.

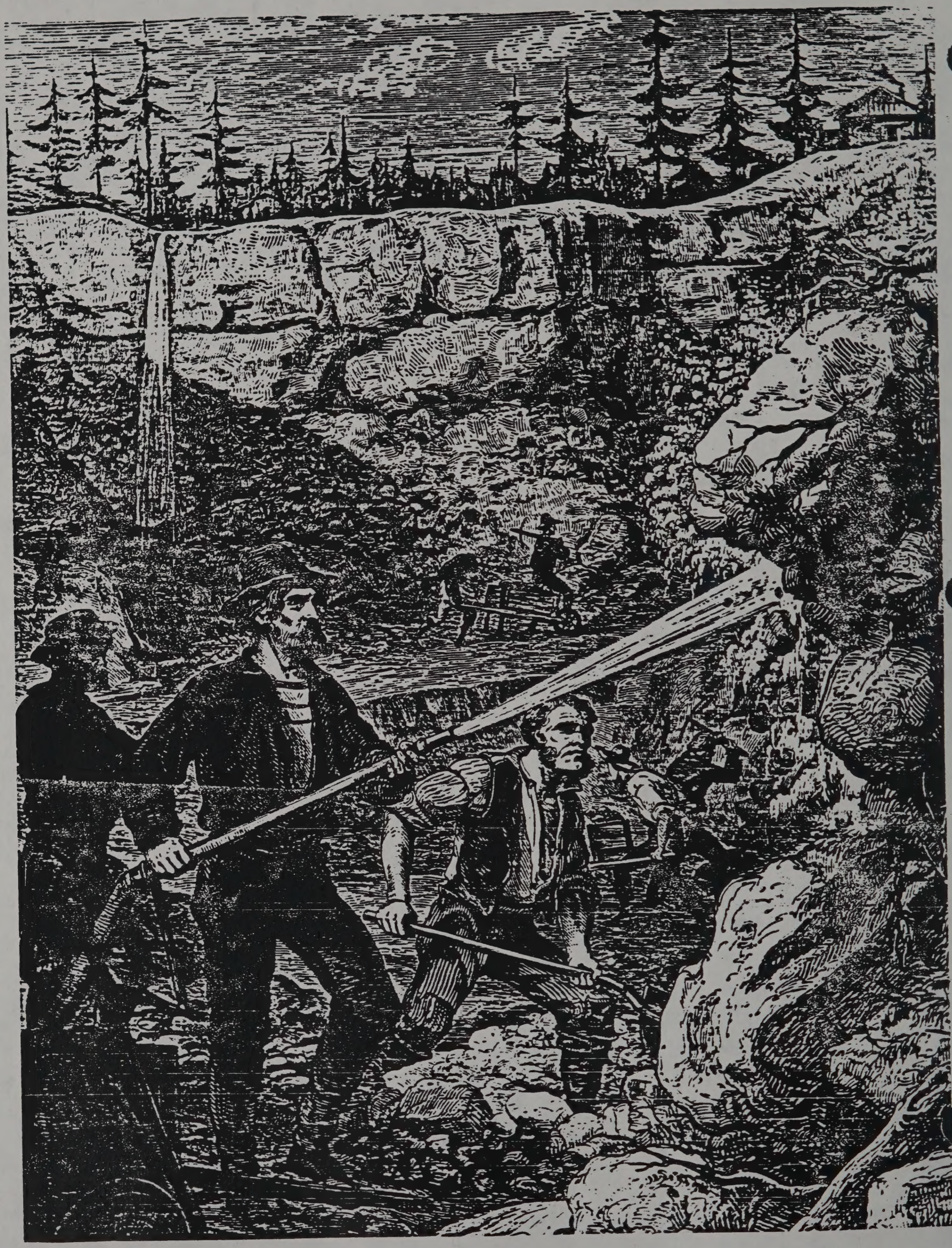
Cretaceous Marine Placers.

11. Cretaceous marine placers, largely in the Chico conglomerate (Upper Cretaceous) beds of northern California. The Lower Cretaceous beds are also reported to contain some gold-bearing layers.

Largest Reserve.

The largest of these possible reserves probably lie in the remaining buried Tertiary stream placers of the northern Sierra Nevada.

The End



Checklist from
Wells (1989)

FIELD GUIDE AND CHECK LIST FOR PLACER INVESTIGATIONS

1. Date of examination ______________________________

2. NAME OF CLAIM(s) OR PROPERTY ______________________________

____

3. State ______________, County ______________ District ______________

4. Township ______________, Range ______________ Section(s) ______________

5. REASON FOR EXAMINATION ______________________________

____

6. EXAMINED BY ______________________________

7. Assisted by ______________________________

8. Others present ______________________________

9. Number of Claims or acres ______________________________

____

10. NAMES OF LOCATORS AND PRESENT OWNER ______________________________

____

____

11. Owner's Address ______________________________

12. TYPE OF DEPOSIT (stream, bench, desert, etc.) ______________________________

____

____

13. Terrain ______________________________

____

14. Gradient of deposit: Less than 5% ( ); More than 5% ( ).

Remarks ______________________________

____

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15. Is the deposit dissected by deep washes or old workings? Yes( ); No ( )

Remarks________________________________

16. Type and extent of overburden________________________

17. Depth to permanent water table________________________

18. Depth to bedrock______________________________

19. Kind of bedrock (rock type)__________________________

20. Hardness of bedrock______________________________

21. Bedrock slope or contour to be expected____________________

22. Are high bedrock pinnacles or reefs in evidence? Yes ( ); No ( )

Remarks________________________________

23. Gravel is Well-rounded ( ); Sub-rounded or Sub-angular ( ); Angular ( )

Remarks________________________________

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24. Does gravel contain rocks over 10-inch ring size? Yes ( ); No ( )

Remarks____

____

25. BOULDERS (Max. size, number, distribution, etc.)______________

____

____

____

26. Rock types noted in gravel______________________________

____

____

____

27. Predominant rock type (if any)__________________________

____

28. SAND (kind, amount, distribution, etc.)__________________

____

____

____

29. Sorting or bedding patterns (if apparent)________________

____

____

____

30. STICKY CLAY? Yes ( ); No ( ). Remarks____________________

____

____

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31. Cemented gravel? Yes ( ); No ( ). Remarks________________________

____

____

32. Caliche? Yes ( ); No ( ). Remarks________________________

____

____

33. Permafrost? Yes ( ); No ( ). Remarks________________________

____

34. Buried timber? Yes ( ); No ( ). Remarks________________________

____

35. Hard or abrasive digging conditions? Yes ( ); No ( ). Remarks________

____

____

____

36. Character of gold:Coarse ( ); Flaky ( ); Fine ( ); Rough ( );Shotty ( );

Smooth ( ); Bright ( ); Stained or coated ( ). Remarks______________

____

____

37. Can good recovery be expected by use of riffles of jigs? Yes ( ); No ( )

Remarks____

____

38. Is recovery said to depend on secret process or special equipment?

Yes ( ); No ( ). Remarks____

____

____

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39. Are black sands said to contain locked gold values? Yes ( ); No ( ).

Remarks____

____

40. Have black sands been checked for valuable minerals other than gold?

Yes ( ); No ( ). Remarks____

____

____

41. Distribution of values in deposit (if known)_________________________

____

____

____

42. Record or evidence of previous sampling______________________________

____

____

____

43. Results of prior sampling (if known)__________________________________

____

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____

44. Are old workings in evidence? Yes ( ); No ( ). Remarks_______________

____

____

45. Past production (if known)____

____

____

____

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46. Date of last production or work

47. Reason for quitting

48. Present work (if any)

49. APPLICABLE MINING METHOD

50. Possible cost to bring property intro production

51. POSSIBLE MINING COST

52. Dimensions of (physically) minable ground

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53. Possible extensions____

54. Maximum yardage indicated to date____

55. Mining equipment on ground____

56. Accessory equipment or improvements on ground____

57. Water supply____

58. Power supply____

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59. DOES PROPERTY HAVE ADEQUATE TAILINGS DUMP ROOM? Yes ( ); No ( ).

Remarks____

____

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____

60. Would mining in this area come under County, State or Federal water quality control regulations? Yes ( ); No ( ). Remarks______________

____

____

____

61. Fish and game regulations? Yes ( ); No ( ). Remarks______________

____

____

62. CAN SETTLING PONDS BE BUILT TO EFFECTIVELY RETAIN OR CLARIFY THE MUDDY WATER? Yes ( ); No ( ). Remarks______________________________

____

____

63. IS PROPERTY SUBJECT TO RESOILING OR OTHER SURFACE RESTORATION REGULATIONS? Yes ( ); No ( ). Remarks______________________________

____

____

____

64. Elevation of property______________________________

65. Climate____

____

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66. Working season_______________

67. Season governed by_______________

____

68. Surface cover and its effect on mining_______________

____

____

69. Merchantable timber or other surface values_______________

____

____

____

70. Nearest town_______________

71. Access_______________

____

____

72. Reference maps_______________

____

____

____

73. Aerial photos (USGS, Forest Service, etc.)_______________

____

____

74. Reference literature_______________

____

____

75. Previous examinations or reports_______________

____

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76. Other reference sources ______________________________

____

____

77. SAMPLING (describe or attach notes) ______________________________

____

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____

____

78. Additional information and remarks ______________________________

____

____

____

____

79. Attach suitable map or sketches (if needed) ______________________________

80. Attach photographs of pertinent features (if available)

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Amalgamation &
Laboratory Excercise

Example Assay Instructions for Placer Concentrates
Mercury Amalgamation

Assay Instructions

Enclosed are six black sand concentrates for mercury amalgamation. You should shortly receive a purchase order to cover the cost. Please do not begin work until it arrives.

The following needs to be done with each sample.

1. Mercury amalgamation:

   A. Report sample dry weight.
   B. Report gold and silver concentration in total milligrams.
   C: Please do not report concentrations in ounces per ton of concentrates.

2. Fineness determination.

3. Return pulps and results to:

   Matt Shumaker
   Bureau of Land Management
   National Training Center
   9828 North 31 Avenue
   Phoenix, AZ 85051.

Please give me a call at (602) 906-5526 if you have any questions.

Example Assay Instructions for Placer Concentrates

Mercury Amalgamation Followed by Cyanide Test

Assay Instructions

Enclosed are 5 black sand concentrates for mercury amalgamation. You should shortly receive a purchase order from BLM Safford District to cover the cost. Please do not begin work until it arrives.

The following needs to be done with each sample.

1. Mercury amalgamation. Report sample dry weight, and gold and silver from amalgamation in total milligrams. *Please do not report concentrations in ounces per ton of concentrates.*
2. Fineness determination.
3. After amalgamation, Cyanide bottle roll. Report gold and silver from cyanide bottle roll in total milligrams. *Please do not report concentrations in ounces per ton of concentrates.*
4. Return pulps and results to:
   Larry Thrasher
   Bureau of Land Management
   Safford District.
   711 14th Avenue
   Safford, AZ 85546.

Please give Matt Shumaker a call at (602) 906-5526 if you have any questions about sample processing.

Please call Larry Thrasher at (520) 428-4040 for purchase order questions.

Mercury Amalgamation of Gold Concentrates

Mercury is not a benign substance. It is toxic and can cause long term health problems. Mercury is also an environmental hazard, and is a regulated hazardous substance. Proper handling is the key to safe, successful mercury amalgamation. Read the mercury metal Material Safety Data Sheet (MSDS). Wear rubber or plastic gloves and safety glasses. Whenever mercury is being handled, work should be done over a large plastic (NOT galvanized!) safety pan to catch any inadvertent spills.

Amalgamation is a means by which the gold and silver in placer gold dissolve in mercury, which is a liquid metal. Mercury will continue to absorb gold and silver until it reaches the consistency of soft butter. The sample must be mechanically agitated for a period of time long enough for the mercury to come into contact with each minuscule piece of placer gold. When amalgamation is completed, the resulting gold, silver, and mercury mixture, known as "amalgam," is dissolved in nitric acid diluted 1:1. The nitric acid dissolves only the silver and mercury, leaving gold in a spongy solid. This is known as "gold sponge."

The resulting gold sponge is weighed and considered as part of the potentially recoverable gold. Most placer gold contains some percentage of silver. This silver is not easily recoverable, but when amalgamations are done commercially, concentration of silver in the parting solution (see below) can be requested in the assay instructions. Placing zinc or copper metal into the parting solution causes mercury to be precipitated out so it can be recovered and reused. A pound of mercury (about equal to one fluid ounce) will last for more than decade if it is properly recycled.

The following steps are from Wells (1969), pp 92-93.

1. Reduce the sample to a black sand concentrate by panning, rocking, or other suitable means.

2. Place the concentrate in a pan or petri dish and manually remove any gold particles which are to be kept in their natural form.

3. Working over a large plastic safety pan, add a globule of clean gold-free mercury about the size of a small bean to the amalgamation vessel.

   Amalgamation can be done in a pan, in a sealed container on a jar mill, or on a gold wheel such as a Goldhound^{R}. Don't forget to work over a safety tub.

   Amalgamate for an hour, or long enough to allow the mercury bead to come into contact with the entire sample.

4. Remove the mercury and place into a 250 ml Pyrex[R] beaker. The mercury may have broken up into many small globules. Removing the mercury is a slow process and will probably require the use of a very fine point eyedropper. Don't become rushed. Take your time and be sure to get every globule, no matter how small.

5. Add about 20 to 30 ml of 1:1 nitric acid and digest until the mercury bead is reduced to the size of a match head. Transfer the bead to a size 0 glazed porcelain parting cup. Pour the leftover parting solution into the stock bottle for later mercury recovery.

   Add fresh acid and complete the digestion using a low setting on a hot plate, if needed. Do not allow the acid to boil! Fine size gold will be left as a tiny coherent sponge-like mass. Pour the leftover parting solution into the stock bottle for later mercury recovery. (Remember: the gold sponge will be really tiny!)

6. After decanting off the acid, using a wash bottle, carefully wash the gold three or more times with distilled water. Decant the used wash water into the parting solution stock bottle.

7. Add a drop or two of alcohol to prevent spattering (70% isopropopyl rubbing alcohol works fine) and dry the parting cup at a low heat.

   The use of a high heat at any of the previous steps can cause spattering which can cause the gold sponge to blow out of the parting cup and disappear, ruining your sample.

8. Over a bunsen burner and under a fume hood or in a very well ventilated area, anneal the gold sponge by bringing the parting cup to a low red heat. This will eliminate any residual mercury and is essential to working with very small gold weights.

9. Transfer the annealed gold to a balance pan and weigh.

Material Safety Data Sheet

MERCURY

May be used to comply with OSHA's Hazard Communication Standard, 29 CFR 1910.1200. Standard must be consulted for specific requirements.

FOR CHEMICAL EMERGENCY
SPILL, LEAK, FIRE, EXPOSURE, OR ACCIDENT
CALL CHEMTREC—DAY OR NIGHT
*800-424-9300
Toll-free in the continental U.S.
*Add long-distance access number if required
483-7616 in District of Columbia
For calls originating outside the Continental U.S.: 202-483-7616—Washington, DC, Collect
ALL CALLS ARE RECORDED

SECTION 1 -

Manufacturer's Name: D.F. GOLDSMITH CHEMICAL & METAL CORP.

Address: 909 Pitner Avenue

City, State, and ZIP: Evanston, IL 60202

Other Information Calls: 708-869-7800

Signature of Person Responsible for Preparation (Optional):

Date Prepared: 9/1/90

SECTION 2 - HAZARDOUS INGREDIENTS/IDENTITY

Hazardous Components(s) (chemical & common name(s))	OSHA PEL	ACGIH TLV	Other Exposure Limits	% (optional)	CAS NO.
MERCURY (METALLIC MERCURY) (QUICKSILVER)	0.05 MG(HG)/M³	0.05 MG(HG)/M³ TWA		100	7439-97-6

SECTION 3 - PHYSICAL & CHEMICAL CHARACTERISTICS

Boiling Point: 675 F (357 C)

Specific Gravity (H_2O=1): 13.6

Vapor Pressure (mm Hg): 0.0012 MMGH @ 20C

Vapor Density (Air=1): 7.0

Solubility in Water: Insoluble

Reactivity in Water: N.A.

Appearance and Odor: Silver-White, Heavy Mobile, Liquid Metal

Melting Point: -38 F (-39 C)

SECTION 4 - FIRE & EXPLOSION DATA

Flash Point F. C.: N/A

Method Used:

Flammable Limits in Air % by Volume: LEL Lower N/A; UEL Upper

Auto-Ignition Temperature: N/A

Extinguisher Media: Dry Chemical, Carbon Dioxide, Water Spray or Foam (1984 Emergency Response Guidebook, DOT P 5800.3)

Special Fire Fighting Procedures: For larger fires, use water spray, fog or alcohol foam (1984 Emergency Response Guidebook, DOT P 5800.3) Firefighting: Move containers from area if possible. Cool containers exposed to flames with water from side until well after fire is out (1984 Emergency Response Guidebook, DOT P 5800.3) Use agents suitable for type of fire;

Unusual Fire and Explosion Hazards: Use water in flooding amounts as a fog. Avoid breathing corrosive and poisonous vapors. Keep upwind.

SECTION 5 - PHYSICAL HAZARDS (REACTIVITY DATA)

Stability: Unstable / Stable

Conditions to Avoid: Does not ignite readily. Flammable, poisonous gases may accumulate in tanks & hopper cars. May ignite combustibles (wood, paper, oil)

Incompatability (Materials to Avoid): Violent Reaction: Acetylinic Compounds; Ammonia; Boron; Diiodophosphide; Ethylene Oxide; Metals (Aluminum; Potassium; Lithium; Sodium; Rubidium); Methyl Azide; Methylsilane; Oxygen; Oxidants (Bromine; Peroxyformic Acid; Chlorine Dioxide; Nitric Acid; Tetracarbonylnickel; Nitromethane; Silver Perchlorate

Hazardous Decomposition Products: Thermal decomposition products include toxic mercury vapors & oxygen.

Hazardous Polymerization: May Occur / Will Not Occur

Conditions to Avoid: None Known

SECTION 6 - HEALTH HAZARDS

Elemental Hg, liquid and vapor, is toxic due to its liquid solubility, lack of charge, and membrane permeability. Inhaled v[illegible] (80%) diffuse rapidly through alveolar membranes into the blood and are systemically transported to body tissues, including the brain. Exposure to high conc. (1.2 mg/m3) of vapors for brief periods can cause pneumonitis, chest pains, dyspnea, coughing.; Later stomatitis, gingivitis, and salivation occur. Hg can be absorbed slowly through the skin. Chronic symptoms involve the CNS with tremors and various neuropsychiatric disturbances. The TLV would be exceeded if the contents of a small Hg clinical thermometer were dispersed in a closed 100' x 100' x 15 room. GI uptake of Hg is low (5%).

FIRST AID:

Eye Contact: Flush with running water for 15 min. including under the eyelids.

Skin Contact: Remove contaminated clothing. Wash affected area with soap and water.

Inhalation: Remove to fresh air. Restore and/or support breathing as needed. Adminster O^2 for chem. pneumonitis.

Ingestion: Gastric lavage with 5% solution of sodium formaldehyde sulfoxylate, followed by 2% $NaHCO_3$, and finally leave 250 cc of the sodium formaldehyde sulfoxylate in the stomach.
Seek medical assistance for further treatment, observation and support.

Skin Contact: Irritant/Sensitizer/Neurotoxin/Nephrotoxin.
Acute Exposure - May cause redness and irritation. Sensitization Dermititis may occur in previously exposed workers. Substance may be absorbed through intact skin causing anuria.

ROUTES OF ENTRY

Eye Contact: Irritant. Acute Exposure - Contact may cause irritation. Solutions are corrosive and may cause corneal injury or burns. Chronic Exposure - Mercury may be deposited in the lens of the eye, causing visual disturbances.

Ingestion: Neurotoxic/Nephrotoxic. Acute Exposure - When ingested, necrosis begins immediately in the mouth, throat, esophagus and stomach. Within a few minutes, violent pain, profuse vomiting, and severe purging may occur. Patient may die within a few minutes from fluid/electrolyte losses and peripheral vascular collapse, but death (from uremia) is usually delayed 5 to 12 days.

Inhalation: Irritant/Sensitizer/Neurotoxin. 28 MG/M3 immediately dangerous to life or health. Acute Exposure - Inhalation of a high concentration of mercury vapor can cause almost immediate dyspnea, cough, fever, nausea and vomiting, diarrhea, stomatitis, salivation and metallic taste. Symptoms may resolve or may progress to necrotizing bronchiolitis, pneumonitis, pulmonary edema, and pneumothorax. This syndrome is often fatal in children. Acidosis and renal damage with renal failure may occur. Inhaling volatile organic mercurials in high concentrations causes metallic taste, dizziness, clumsiness, sl[illegible] speech, diarrhea, and sometimes, fatal convulsions. Chronic Exposure - Inhalation of mercury vapor, dusts, over a long p[illegible] causes mercurialism. Findings extremely variable & include tremors, salivation, stomatitis, loosening of teeth, blue lines on gums, pain & numbness in extremities, nephritis, diarrhea, anxiety, headache, weight loss, anorexia, mental depression, insomnia, irritability & instability, hallucinations and evidence of mental deterioration.

SECTION 7 - SPECIAL PRECAUTIONS AND SPILL/LEAK PROCEDURES

Store in closed unbreakable containers (polyethylene) in a cool, dry, well-ventilated area away from sources of heat. Protect containers from physical damage.

Mercury evaporates very slowly. Spilled Hg forms many tiny globules that will evaporate faster than a single pool and can develop a significant concentration of vapors in an unventilated area. Such vapors can be poisonous, especially if breathed over a long period of time. Heated Hg evolves high levels of toxic vapors.

DO NOT TOUCH SPILLED MATERIAL. STOP LEAK IF YOU CAN DO IT WITHOUT RISK. FOR SMALL SPILLS, TAKE UP WITH SAND OR OTHER ABSORBENT MATERIAL AND PLACE INTO CONTAINERS FOR LATER DISPOSAL. A MERCURY SPILL KIT MAY ALSO BE USED FOR SMALL SPILLS IN THE WORKPLACE. FOR LARGER SPILLS, DIKE FAR AHEAD OF SPILL FOR LATER DISPOSAL. KEEP UNNECESSARY PEOPLE AWAY. ISOLATE HAZARD AREA AND DENY ENTRY.

SECTION 8 - SPECIAL PROTECTION INFORMATION/CONTROL MEASURES

Provide adequate exhaust ventilation to meet TLV requirements in the workplace. Operations requiring an Hg surface should reduce the temp. of Hg to limit vaporization and minimize vapor exposure by using a local exhaust.

Self-contained breathing apparatus can be used up to 5 mg/m^3 with a full facepiece above 1 mg/m^3. Positive pressure-type air supplied breathing equipment has been recommended above 5 mg/m^3.

Avoid eye contact by use of chemical safety glasses. Wear rubber gloves and protective clothing appropriate for the work situation. Separate work and street clothing. Store work clothing in special lockers. Showers to be taken before changing to street clothes.

Provide preplacement and periodic medical exams for those regularly exposed to Hg, with emphasis directed to CNS, skin, lungs, liver, kidneys and G.I. tract.

DIVISION OF ASHLAND OIL, INC.
P.O. BOX 2219, COLUMBUS, OHIO 43216 • (614) 889-3333

Ashland

001977 NITRIC ACID REAG ACS 4X7# CS

ACCEPTED BY O.S.H.A. AS ESSENTIALLY SIMILIAR TO O.S.H.A. FORM 20

24-HOUR EMERGENCY TELEPHONE: 606-324-1133 (LOCATED AT ASHLAND, KENTUCKY)

ASHLAND PRODUCT NAME: NITRIC ACID REAG ACS 4X7# CS

UNIVERSAL SCIENTIFIC
1004A E. VISTA DEL CERRO
TEMPE, AZ 85281

05 50 026 9315600-002
DATA SHEET NO: 0000998-003
LATEST REVISION DATE: 11/77-77333
PRODUCT: 3657205
INVOICE: 511240
INVOICE DATE: 01/31/84
TO:

ATTN: PURCHASING/SAFETY DEPT.

SECTION I-PRODUCT IDENTIFICATION

GENERAL OR GENERIC ID: INORGANIC ACID

HAZARD CLASSIFICATION: (12) CORROSIVE (173.240) AND OXIDIZER (173.151)

SECTION II-HAZARDOUS COMPONENTS

INGREDIENT	PERCENT	PEL	TLV	#
NITRIC ACID	70	2	2 PPM	

SECTION III-PHYSICAL DATA

PROPERTY	REFINEMENT	MEASUREMENT
INITIAL BOILING POINT	FOR PRODUCT	250.00 DEG F (121.11 DEG C) @ 760.00 MMHG
VAPOR PRESSURE	FOR PRODUCT	5.50 MMHG @ 68.00 DEG F (20.00 DEG C)
VAPOR DENSITY	AIR = 1	1.3
SPECIFIC GRAVITY		1.413 @ 68.00 DEG F (20.00 DEG C)
PERCENT VOLATILES		100.00%
EVAPORATION RATE		SLOWER THAN ETHER

SECTION IV-FIRE AND EXPLOSION DATA

FLASH POINT NOT APPLICABLE

EXPLOSIVE LIMIT NOT APPLICABLE

EXTINGUISHING MEDIA: WATER FOG

HAZARDOUS DECOMPOSITION PRODUCTS: MAY FORM TOXIC MATERIALS:, NITROGEN COMPOUNDS, ACID FUMES

SPECIAL FIREFIGHTING PROCEDURES: SELF-CONTAINED BREATHING APPARATUS WITH A FULL FACEPIECE OPERATED IN PRESSURE-DEMAND OR OTHER POSITIVE PRESSURE MODE AND FULL BODY PROTECTIVE CLOTHING.

UNUSUAL FIRE & EXPLOSION HAZARDS: ACID REACTS WITH MOST METALS TO RELEASE HYDROGEN GAS WHICH CAN FORM EXPLOSIVE MIXTURES WITH AIR.

SECTION V-HEALTH HAZARD DATA

PERMISSIBLE EXPOSURE LEVEL 2 PPM

THRESHOLD LIMIT VALUE 2 PPM

EFFECTS OF OVEREXPOSURE: FOR PRODUCT

EYES - CAUSES SEVERE DAMAGE AND EVEN BLINDNESS VERY RAPIDLY.
SKIN - CAUSES BURNS, POSSIBLE DEEP ULCERATION.
BREATHING - OF FUMES CAN CAUSE DAMAGE TO NASAL AND RESPIRATORY PASSAGES.
SWALLOWING - RESULTS IN SEVERE DAMAGE TO MUCOUS MEMBRANES AND DEEP TISSUES.

FIRST AID:

IF ON SKIN: IMMEDIATELY FLUSH EXPOSED AREA WITH WATER FOR AT LEAST 15 MINUTES. GET MEDICAL ATTENTION. REMOVE CONTAMINATED CLOTHING. LAUNDER CONTAMINATED CLOTHING BEFORE RE-USE.
DISCARD CONTAMINATED SHOES.

IF IN EYES: IMMEDIATELY FLUSH WITH LARGE AMOUNTS OF WATER FOR AT LEAST 15 MINUTES, LIFTING UPPER AND LOWER LIDS OCCASIONALLY. GET IMMEDIATE MEDICAL ATTENTION.
IF PHYSICIAN IS NOT IMMEDIATELY AVAILABLE, CONTINUE FLUSHING WITH WATER.
DO NOT USE CHEMICAL ANTIDOTE.

IF SWALLOWED: DO NOT INDUCE VOMITING. VOMITING WILL CAUSE FURTHER DAMAGE TO

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SECTION V-HEALTH HAZARD DATA (CONTINUED)

THE THROAT. DILUTE BY GIVING WATER. GIVE MILK OF MAGNESIA. KEEP WARM, QUIET. GET MEDICAL ATTENTION IMMEDIATELY.

IF BREATHED: IF AFFECTED, REMOVE INDIVIDUAL TO FRESH AIR. IF BREATHING IS DIFFICULT, ADMINISTER OXYGEN. IF BREATHING HAS STOPPED GIVE ARTIFICIAL RESPIRATION. KEEP PERSON WARM, QUIET AND GET MEDICAL ATTENTION.

SECTION VI-REACTIVITY DATA

HAZARDOUS POLYMERIZATION: CANNOT OCCUR

STABILITY: STABLE

INCOMPATABILITY: AVOID CONTACT WITH:, STRONG ALKALIES., ORGANIC MATERIALS, REDUCING AGENTS

SECTION VII-SPILL OR LEAK PROCEDURES

STEPS TO BE TAKEN IN CASE MATERIAL IS RELEASED OR SPILLED:

SMALL SPILL: COVER THE CONTAMINATED SURFACE WITH SODIUM BICARBONATE OR A SODA ASH/SLAKED LIME MIXTURE (50-50). MIX AND ADD WATER IF NECESSARY TO FORM A SLURRY. SCOOP UP SLURRY AND WASH SITE WITH SODA ASH SOLUTION.

LARGE SPILL: ELIMINATE ALL IGNITION SOURCES (FLARES, FLAMES INCLUDING PILOT LIGHTS, ELECTRICAL SPARKS). PERSONS NOT WEARING PROTECTIVE EQUIPMENT SHOULD BE EXCLUDED FROM AREA OF SPILL UNTIL CLEAN-UP HAS BEEN COMPLETED. STOP SPILL AT SOURCE, DIKE AREA OF SPILL TO PREVENT SPREADING, PUMP LIQUID TO SALVAGE TANK. REMAINING LIQUID MAY BE TAKEN UP ON SAND, CLAY, EARTH, FLOOR ABSORBENT, OR OTHER ABSORBENT MATERIAL AND SHOVELED INTO CONTAINERS.

WASTE DISPOSAL METHOD:

SMALL SPILL: FLUSH DOWN DRAIN WITH LARGE AMOUNTS OF WATER IN ACCORDANCE WITH APPLICABLE REGULATIONS.

LARGE SPILL: COLLECT AND ADD SLOWLY TO LARGE VOLUME OF AGITATED SOLUTION OF SODA ASH AND SLAKED LIME. ADD NEUTRALIZED SOLUTION TO EXCESS RUNNING WATER IN ACCORDANCE WITH APPLICABLE REGULATIONS.

SECTION VIII-PROTECTIVE EQUIPMENT TO BE USED

RESPIRATORY PROTECTION: IF TLV OF THE PRODUCT OR ANY COMPONENT IS EXCEEDED, A NIOSH/MSHA JOINTLY APPROVED AIR SUPPLIED RESPIRATOR IS ADVISED IN ABSENCE OF PROPER ENVIRONMENTAL CONTROL. OSHA REGULATIONS ALSO PERMIT OTHER NIOSH/MSHA RESPIRATORS UNDER SPECIFIED CONDITIONS. (SEE YOUR SAFETY EQUIPMENT SUPPLIER). ENGINEERING OR ADMINISTRATIVE CONTROLS SHOULD BE IMPLEMENTED TO REDUCE EXPOSURE.

VENTILATION: PROVIDE SUFFICIENT MECHANICAL (GENERAL AND/OR LOCAL EXHAUST) VENTILATION TO MAINTAIN EXPOSURE BELOW TLV(S).

PROTECTIVE GLOVES: WEAR RESISTANT GLOVES SUCH AS:, NEOPRENE, POLYVINYL CHLORIDE

EYE PROTECTION: CHEMICAL SPLASH GOGGLES AND FACE SHIELD (8" MIN.) IN COMPLIANCE WITH OSHA REGULATIONS ARE ADVISED; HOWEVER, OSHA REGULATIONS ALSO PERMIT OTHER TYPE SAFETY GLASSES. (CONSULT YOUR SAFETY EQUIPMENT SUPPLIER)

OTHER PROTECTIVE EQUIPMENT: TO PREVENT SKIN CONTACT, WEAR IMPERVIOUS CLOTHING AND BOOTS.

SECTION IX-SPECIAL PRECAUTIONS OR OTHER COMMENTS

ADDITION TO WATER RELEASES HEAT WHICH CAN RESULT IN VIOLENT BOILING AND SPATTERING. ALWAYS ADD SLOWLY AND IN SMALL AMOUNTS. NEVER USE HOT WATER.

CONTAINERS OF THIS MATERIAL MAY BE HAZARDOUS WHEN EMPTIED. SINCE EMPTIED CONTAINERS RETAIN PRODUCT RESIDUES (VAPOR, LIQUID, AND/OR SOLID), ALL HAZARD PRECAUTIONS GIVEN IN THE DATA SHEET MUST BE OBSERVED.

THE INFORMATION ACCUMULATED HEREIN IS BELIEVED TO BE ACCURATE BUT IS NOT WARRANTED TO BE WHETHER ORIGINATING WITH ASHLAND OR NOT. RECIPIENTS ARE ADVISED TO CONFIRM IN ADVANCE OF NEED THAT THE INFORMATION IS CURRENT, APPLICABLE, AND SUITABLE TO THEIR CIRCUMSTANCES.

MERCURY (Metal)

QUICKSILVER

EXCEPTIONAL CONTACT HAZARD - READ MATERIAL SAFETY DATA SHEET. MAY BE FATAL IF SWALLOWED OR INHALED. EMITS TOXIC VAPORS, ESPECIALLY WHEN HEATED.

Do not get in eyes, on skin, on clothing. Do not breathe dust. Keep in tightly closed container. Use with adequate ventilation. Wash thoroughly after handling.

EFFECTS OF OVEREXPOSURE: Inhalation of vapors may cause coughing, chest pains, nausea and vomiting. Chronic effects of overexposure may include kidney and/or liver damage and central nervous system depression. Chronic effects of mercury poisoning include a buildup of the metal in the brain, liver and kidneys. Symptoms include headache, tremors, loose teeth, loss of appetite, blisters on the skin and impaired memory.

FIRST AID PROCEDURES: If swallowed, if conscious, immediately induce vomiting. If inhaled, remove to fresh air. If not breathing, give artificial respiration. If breathing is difficult, give oxygen. In case of contact, immediately flush eyes or skin with plenty of water for at least 15 minutes while removing contaminated clothing and shoes. Wash clothing before re-use.

Consult MSDS for further hazardous information and instructions. CAS NO. [7439-97-6]

NITRIC ACID

POISON! DANGER!

LIQUID AND VAPOR CAUSE SEVERE BURNS - MAY BE FATAL IF SWALLOWED. HARMFUL IF INHALED AND MAY CAUSE DELAYED LUNG INJURY. STRONG OXIDIZER - CONTACT WITH OTHER MATERIAL MAY CAUSE FIRE. SPILLAGE MAY CAUSE FIRE OR LIBERATE DANGEROUS GAS.

Keep from contact with clothing and other combustible materials. Do not get in eyes, on skin, on clothing. Do not breathe vapor. Keep in tightly closed container. Use only with adequate ventilation. Wash thoroughly after handling. When handling, use proper protective equipment.

EFFECTS OF OVEREXPOSURE: Liquid may cause severe burns to skin and eyes. Inhalation of vapors may cause severe irritation of the respiratory system. Inhalation of vapors may cause coughing, chest pains, difficulty breathing, or unconsciousness. Ingestion may be fatal.

FIRST AID PROCEDURES: If swallowed, do NOT induce vomiting. Give water, milk, or milk of magnesia. If inhaled, remove to fresh air. If not breathing, give artificial respiration. If breathing is difficult, give oxygen. In case of contact, immediately flush eyes or skin with plenty of water for at least 15 minutes while removing contaminated clothing and shoes. In all cases, contact a physician.

Consult MSDS for further health and safety information. CAS NO. [7697-37-2]

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Mercury Spill Cleanup Kit

Material Safety Data Sheet

Hg Abso MSDS 1609

Identity (Trade Name As Used On Label)

Lab Safety Supply, Inc.
Manufacturer

P.O. Box 1368
Address

Janesville, WI 53547-1368

608-754-2345
Phone Number (For Information)

608-754-2345
Emergency Phone Number

Telex*

N/A
MSDS Number*

N/A
CAS Number*

5-16-90
Date Prepared

N/A
Prepared By*

Note: Blank spaces are not permitted. If any item is not applicable, or no information is available, the space must be marked to indicate that.

SECTION 1 - MATERIAL IDENTIFICATION AND INFORMATION

COMPONENTS — Chemical Name & Common Names (Hazardous Components 1% or greater; Carcinogens 0.1% or greater)	%*	OSHA PEL	ACGIH TLV	OTHER LIMITS RECOMMENDED
Granular Metal (Non-regulated) *	94-96		N/A	N/A
		$5mg/m^3$	Respirable Dust	
		$15mg/m^3$	Total Dust	
Carboxylic Acid *	4-6		N/A	N/A
		$5mg/m^3$	Respirable Dust	
		$15mg/m^3$	Total Dust	
* Above components are a Trade Secret.				
Non-Hazardous Ingredients				
TOTAL	100			

SECTION 2 - PHYSICAL / CHEMICAL CHARACTERISTICS

Boiling Point: N/A

Specific Gravity (H_2O = 1): Bulk Density 2.35gm/cc

Vapor Pressure (mm Hg and Temperature): N/A

Melting Point: N/A

Vapor Density (Air = 1): N/A

Evaporation Rate (______ = 1): N/A

Solubility in Water: Moderate

Water Reactive: Hydrogen may be released

Appearance and Odor: Gray granular solid

SECTION 3 - FIRE AND EXPLOSION HAZARD DATA

Flash Point and Method Used: N/A

Auto-Ignition Temperature: N/A

Flammability Limits in Air % by Volume: N/A — LEL: N/A — UEL: N/A

Extinguisher Media: CO_2 or dry chemical (Do not use water - water will react violently with molten materi

Special Fire Fighting Procedures: None

Unusual Fire and Explosion Hazards: None

*Optional

SECTION 4 - REACTIVITY HAZARD DATA

STABILITY ☒ Stable ☐ Unstable	Conditions To Avoid	N/A
Incompatability (Materials to Avoid)	oxidizing agents, acids, bases, halogens, reducing agents, amines, chlorinated solvents, sulfur	
Hazardous Decomposition Products	May release carbon monoxide, carbon dioxide	
HAZARDOUS POLYMERIZATION ☐ May Occur ☒ Will Not Occur	Conditions To Avoid	N/A

SECTION 5 - HEALTH HAZARD DATA

PRIMARY ROUTES OF ENTRY: ☒ Inhalation ☐ Ingestion ☐ Skin Absorption ☐ Not Hazardous

CARCINOGEN LISTED IN: ☐ NTP ☐ OSHA ☐ IARC Monograph ☒ Not Listed

HEALTH HAZARDS

Acute: irritation to eyes, nose, throat, metallic taste in mouth, flu like symptoms.

Chronic: chromosomal anomalies in Leukocytes reported, Arthritic lameness and inflammation of gastrointestinal tract reported in animal studies.

Signs and Symptoms of Exposure: Irritation of nose, throat and eyes

Medical Conditions Generally Aggravated by Exposure: N/A

EMERGENCY FIRST AID PROCEDURES - Seek medical assistance for further treatment, observation and support if necessary.

Eye Contact: Flush eyes with plenty of water for at least 15 minutes

Skin Contact: Flush with plenty of water

Inhalation: Remove to fresh air, seek medical attention immediately

Ingestion: Dilute with plenty of water seek medical attention immediately

SECTION 6 - CONTROL AND PROTECTIVE MEASURES

Respiratory Protection (Specify Type): Use nuisance dust mask at dust levels over 15mg/m^3, 3M 9908 or equivalent may be desirable for Mercury clean up procedures

Protective Gloves: Polyethylene, rubber or PVC

Eye Protection: Safety glasses recommended

VENTILATION TO BE USED: ☐ Local Exhaust ☒ Mechanical (general) ☐ Special ☐ Other (specify)

Other Protective Clothing and Equipment: None required in normal use

Hygienic Work Practices: Wash hands after use

SECTION 7 - PRECAUTIONS FOR SAFE HANDLING AND USE / LEAK PROCEDURES

Steps to be Taken If Material Is Spilled Or Released: Sweep up material and place in waste disposal container

Waste Disposal Methods: Mercury free material may be containerized and disposed of in licensed landfill Dispose of mercury containing material in accordance with local, State and Federal regulations for hazardous waste

Precautions to be Taken in Handling and Storage: Store in cool dry place, away from incompatable materials

Other Precautions and/or Special Hazards: N/A

NFPA Rating*: Health 0 Flammability 1 Reactivity 1 Special ____

HMIS Rating*: Health ____ Flammability ____ Reactivity ____ Personal Protection ____

*Optional

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MATERIAL SAFETY DATA SHEET

MSDS1014

AMEREX CORPORATION
POST OFFICE BOX 81
TRUSSVILLE, ALABAMA 35173-0081

HALON® Fire Extinguishers

DATE PREPARED: 2/10/89

SUPERSEDES: 6/22/88

RESPONSIBLE PERSONS: SAFETY AND HEALTH DEPARTMENT
TELEPHONE: (205)655-3271

IDENTIFICATION

PRODUCT NAME: HALON 1211
SYNONYMS: Bromochlorodifluoromethane, BCF

HAZARDOUS INGREDIENTS

MATERIAL	CAS NO.	OSHA PEL	ACGIH TLV
Bromochlorodifluoromethane	353-59-3	Unknown	Not listed

OTHER INGREDIENTS

MATERIAL	CAS NO.
None	

PHYSICAL AND CHEMICAL CHARACTERISTICS

BOILING POINT (F): 26	SPECIFIC GRAVITY (H_2O=1): 1.83 (liquid)	VAPOR PRESSURE(MM HG): 778 @ 60 F
PERCENT VOLATILE (%): 100	VAPOR DENSITY(AIR = 1): 5.7	EVAPORATION RATE: High
SOLUBILITY IN WATER: None	REACTIVITY IN WATER: NA	MELTING POINT (F): NA
pH: No data	APPEARANCE & ODOR: Colorless gas and liquid with very faint, sweet odor.	
FLASH POINT(F): None	AUTO IGNITION TEMPERATURE (F): NA	FLAMMABLE LIMITS IN AIR BY VOL.: NA

EXTINGUISHER MEDIA: None. This material is an extinguishing agent.

SPECIAL FIRE FIGHTING PROCEDURES: Self-contained breathing apparatus with full facepiece and protective clothing when re-entering unventilated fire areas where product has been used.

UNUSUAL FIRE AND EXPLOSION HAZARDS: When Halon 1211 is discharged onto a fire, it decomposes above 900 F, releasing bromide ions, the extinguishing agent. Halogen compounds, such as halogen acids, are also formed. These by-products, although harmful if inhaled, are easily detected. Only a few PPM create an unpleasant, acrid odor which serves as a warning to the user. After the extinguisher is discharged, the area should be vacated until ventilation clears the atmosphere.

PHYSICAL HAZARDS

STABILITY: Stable — CONDITIONS TO AVOID: Decomposes under fire conditions

INCOMPATIBILITY (MATERIALS TO AVOID): Active metals such as powdered alumina and magnesium and fires of metal hydrides

HAZARDOUS DECOMPOSITION PRODUCTS: Decomposes above 900 to give free halogens, halogen acid and small amounts of carbonyl halides.

HAZARDOUS POLYMERIZATION: Will not occur

CONDITIONS TO AVOID: NA

HEALTH EFFECTS AND FIRST AID

EFFECTS OF ACUTE OVEREXPOSURE FOR PRODUCT:

EYE	The liquid form of this material can produce chilling sensations and discomfort.
SKIN:	Evaporation of liquid from the skin can produce chilling sensations.
BREATHING:	Exposures for more than a few minutes above 4% (17 lbs/1000 cu. ft. air) concentration can produce dizziness, impaired coordination, and cardiac effects.
SWALLOWING:	Not likely to occur since this material is a gas at room temperature.

FIRST AID:

IF IN EYES:	Flush with water for 15 minutes. If redness, itching, or a burning sensation develops, have eyes examined and treated by medical personnel.
IF ON SKIN:	Wash with soap and water. If redness, itching, or a burning sensation develops, seek medical attention.
IF BREATHED:	Remove victim to fresh air. If cough or other respiratory symptoms develop, consult medical personnel.
IF SWALLOWED:	Give 1 or 2 glasses of warm water to drink and seek medical attention.

PRIMARY ROUTES OF ENTRY: Eyes, Skin Contact, Breathing

MEDICAL CONDITIONS GENERALLY AGGRAVATED BY EXPOSURE: In susceptible individuals, cardiac sensitization to circulating epinephrine compounds can result in potentially fatal heart arrhythmias.

EFFECTS OF CHRONIC OVEREXPOSURE FOR PRODUCT: Unknown

PRODUCT LISTED AS CARCINOGEN OR POTENTIAL CARCINOGEN:

NATIONAL TOXICOLOGY PROGRAM	[ ] YES [X] NO	IARC MONOGRAPHS	[ ] YES [X] NO	OSHA	[ ] YES [X] NO

CONTROL MEASURES AND PROTECTIVE EQUIPMENT

RESPIRATORY PROTECTION (TYPE): Not normally needed. If needed, use MSHA/NIOSH approved respirator for organic vapors.

VENTILATION: GENERAL AREA: Recommended LOCAL EXHAUST: Recommended

SKIN PROTECTION: Impervious Gloves for Liquid Exposure EYE PROTECTION: Safety Glasses

OTHER PROTECTIVE CLOTHING OR EQUIPMENT: Eye wash station and safety shower in work area when working with liquified product.

WORK/HYGIENE PRACTICES: Use good personal hygiene and good housekeeping practices.

SPECIAL PRECAUTIONS AND SPILL/LEAK PROCEDURES

PRECAUTIONS TO BE TAKEN IN HANDLING AND STORAGE: Store in a cool area with good ventilation. Keep vapors away from high temperature surfaces to avoid toxic and corrosive decomposition products.

OTHER PRECAUTIONS: Enforce "No Smoking" rules in areas of use.

STEPS TO BE TAKEN IN CASE PRODUCT IS RELEASED OR SPILLED: Ventilate spill area and recover any liquid.

HMIS RATINGS:

HEALTH 2

HMIS HAZARD INDEX

MINIMAL	0
SLIGHT	1

McCulloch's
Mine Design

EMPTY SECTION

Hustrulid

Underground Mining Methods Handbook

W.A. Hustrulid

Editor

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ORE RESERVE CALCULATION

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INTRODUCTION

The estimation of ore reserves is a process that begins with the earliest exploration stages on a property and continues throughout any subsequent evaluation and exploitation of the deposit. During exploration and preliminary evaluation, the results of these reserve estimates constitute the basic data for prefeasibility studies and economic analysis. The decision to continue exploration and development or to abandon a prospect is often based upon these studies.

During the active life of a mine, reserve computations are continuously revised to assist in development planning, cost and efficiency analyses, quality control, and improvement of extraction and processing methods. Accurate reserve estimates are also required when financing a project, purchasing or selling a property, and for accounting purposes such as depletion and tax calculation.

It is important to remember that the reliability of ore reserve estimates varies progressively through time as more and more information becomes available. The lowest order of reliability of estimation of reserves exists at the time of discovery. The maximum level of certainty concerning the ore reserves within a deposit is reached when the deposit is completely mined out. Between these two extremes are variable levels of certainty as to the tonnage and grade of the resource. This is particularly true of that portion of the mineral resource that constitutes the "minable reserve," as this portion is dependent upon economic as well as geological and technological factors.

In the following discussion, several of the factors affecting ore reserve computation and some of the commonly used methods of calculation are presented. The first part of the discussion is confined to classical methods of hand calculation utilizing level maps and sections. The second part of the discussion presents an overview of somewhat more sophisticated methods of geostatistics which have been developed within recent years. Geostatistical methods frequently provide the best ore reserve estimates for many deposits. However, a digital computer is required for such geostatistically derived estimates due to the number and complexity of the computations involved. For many properties the classical nongeostatistical methods are adequate, particularly in the early exploration stages. In fact, it is the author's contention that even at those properties where the geostatistical approach is employed for reserve estimation, the geological and engineering staffs should prepare an estimate by traditional means. This provides an internal audit for the reserves and at the same time requires a continuing close appraisal of geologic problems that influence the presently known geostatistically estimated reserves and production at the mine.

CLASSIFICATION OF ORE RESERVES

The definition and classification of ore reserves has varied over the period of development of the modern mining industry. "Ore" is generally understood to be any naturally occurring, in-place, mineral aggregate containing one or more valuable constituents which may be recovered at a profit under existing economic conditions. This definition ignores special situations, such as wartime production, or those cases when an otherwise unprofitable deposit may be exploited for political or social reasons.

Ore reserves are classified with respect to the confidence level of the estimate. Traditionally, ore reserves have been classified as proven (measured), probable (indicated), possible, and inferred. Historically, proven ore has been regarded as that which is "blocked out," i.e., measured, sampled, and assayed on four sides; probable ore as blocked on three sides; possible ore as blocked on two sides; and inferred ore as ore grade material that is known on only one side.

More recently, the US Bureau of Mines (USBM) introduced the following ore reserve classification:

Measured Ore

Measured ore is ore for which tonnage is computed from dimensions revealed in outcrops, trenches, workings, and drill holes, and for which the grade is computed from the results of detailed sampling. The sites for inspection, sampling, and measurement are so closely spaced and the geologic character is so well-defined that the size, shape, and mineral content are well established. The computed tonnage and grade are judged to be accurate within limits which are stated, and no such limit is judged to differ from the computed tonnage or grade by more than 20%.

Indicated Ore

Indicated ore is ore for which tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, sampling, and measurement are too widely or otherwise inappropriately spaced to outline the ore completely or to establish its grade throughout.

Inferred Ore

Inferred ore is ore for which quantitative estimates are based largely on a broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. These estimates are based on an assumed continuity or repetition for which there is geologic evidence: this evidence may include comparison with deposits of similar type. Mineral bodies that are completely concealed may be included if there is specific geologic evidence of their presence. Specific es-

timates of inferred ore usually include a statement of the special limits within which the inferred ore may occur.

The traditional classification system has been particularly applicable to tabular vein or bedded-type deposits. This results from normal mine practice for such deposits, which is to develop the deposit by a system of rectangular blocks, the boundaries of which are defined by drifts and raises. Prior to extraction, the vein is exposed on all sides and may be sampled in detail. In more recent years, the underground mining of large deposits of indefinite shape, such as porphyry copper deposits, has led to an increasing reliance on diamond drill-hole data to define ore continuity. Prior to mining of these deposits, however, it is still common practice to do a limited amount of underground development. This underground development serves to obtain sample information, including large bulk samples for metallurgical testing, as well as provide additional detailed geological and engineering data for ore reserve estimation and verification.

By the time a deposit is ready for development, there usually exist two ore reserve estimates: a geologic reserve or total resource estimate, and a mining ore reserve. The geologic reserve is an estimate including all known mineralization above a certain grade within the deposit. However, the geologic reserve figure may or may not be associated with a specific mining cutoff grade. The mining reserve constitutes that portion of the geologic reserve that can be mined and processed at a profit. The mining reserve is always less than or equal to the geologic reserve estimate. This results because a variable proportion of the ore body must be left unmined for a variety of reasons. These reasons include the need for pillars for ground support, metallurgical problems, width of mineralization, or other economic and engineering factors.

ORE RESERVE PARAMETERS

An ore reserve estimate contains two important parameters: the amount of ore and the average grade or value of that ore. In metal mines, the amount of ore is usually expressed in either metric tons (1000 kg) or short tons (2000 lb). Grades are normally expressed as a percentage for base metal ores, whereas precious metals may be reported as troy ounces per ton, pennyweights per ton, or grams/metric ton. The reporting system used depends upon the geographic location of the deposit and/or company policy. It is also fairly common in old reports to find gold and silver values stated in dollars. This practice makes the utilization of this data difficult unless the conversion prices are given in the report. Another doubtful practice, common in some countries, is to state reserves simply as contained metal, i.e., pounds of uranium oxide or fine tons of tin. While these figures are sometimes useful for certain economic analyses, they provide little useful engineering information and constitute a resource statement rather than an ore reserve estimate.

The calculation of the tonnage and grade of a deposit requires the collection and documentation of a considerable amount of data. These data include accurate assay information, plans and sections, details of ore controls, the tonnage factor, applicable cutoff grade to be used, potential mineral recovery, and engineering details such as minimum mining width and anticipated dilution. Each of these items is discussed in the following sections.

Grade Determination

The average grade of an ore deposit or of a specific block within a deposit is based on assays of samples collected within the block or deposit. The collection, preparation, and analysis of these samples is often the single most critical operation in evaluation of the reserves for a mineral property. Sampling theory and practice constitutes a complex subject in its own right and only some of the more important points are touched upon in the following summary. (For further information, the following references should be consulted: Forrester, 1946; McKinstry, 1948; Gy, 1968; Cummins and Given, 1973; Koch and Link, 1970, 1974; David, 1977; Barnes, 1980).

Cutoff Grade: Associated with the definition of ore grade is the concept of cutoff grade. The cutoff grade is the minimum ore grade which can be mined at a profit under economic conditions existing at a particular point in time. The cutoff grade can vary with time due to changes in such factors as commodity prices, operating costs, and taxes. The cutoff grade used for any reserve calculation should always be stated.

Sampling: Sampling of an ore deposit is a process of approximation. The objective is to arrive at an average value for the samples which most closely represents the true average value for the body in question. The importance of attention to detail during the sampling program becomes apparent when it is realized that in the case of a very well-sampled block of ore from a vein-type deposit, the actual sample volume may only represent about 0.25% of the block. In other cases, such as the sampling of a porphyry copper deposit by diamond drilling, the sample volume may constitute only about 0.004% of the ore body. In order to obtain the most accurate grade estimation, it is imperative that the sampling crews and procedures be carefully monitored by members of the geological and engineering staff.

Channel Sampling: The classical method of sampling an ore deposit consists of cutting a relatively precise channel of constant depth and width across the exposed width of the vein. These samples may be cut with a hammer and chisel or air hammer, and the chips collected on a canvas sheet spread on the floor of the working. The samples are collected across the full width of the vein, or at some uniform fixed length in wide or indefinite zones. In complex veins, any identifiable subdivisions should be sampled separately. Samples are collected in a consistent manner throughout the deposit. These may be taken at right angles to the contacts and dip of the vein, or if this is impractical, measured sample vein widths can be corrected to true widths by simple trigonometry. However, in regular dipping bodies the sample can be collected as a horizontal channel across the zones since vertical height of projection times horizontal width is equal to true width times true length. The important thing is that the samples are collected in a consistent manner and that any local variation or change in the sampling practice is recorded. If possible, samples should be collected at regular intervals along the working. Samples can be taken from either the face, rib, or back of the workings.

Preferably the working area to be sampled should be

washed down or at least brushed clean before sampling begins. This is done to reduce the potential for contamination of the sample by muck and loose fragments on the face being sampled. The sample area should be chipped clean and rough projections removed.

The location of the sample is then marked on the face, rib, or back using a can of spray paint or chalk line. If the sample channel is to be broken into several parts representing distinctly different portions of the vein or mineralization, these should also be marked on the face. A drawing and, if possible, a photograph of the area being sampled provide additional documentation. This is particularly important where the sample is collected from a surface that will be mined or covered by support materials. Such records are of considerable value when it becomes necessary to reinterpret the correlation of mineralization if the sample site has been removed from access by other mining activity.

Following marking of the sample site, a clean canvas tarp is spread beneath the sample area. The sample is then chipped out along the marked line and the fragments and fine material collected on the tarp. When the sample cutting has been completed, the material is homogenized by "rolling" the sample back and forth on the tarp. The sample is then split, if necessary, and put into plastic or cloth bags. Sample numbers and tags are then prepared and placed inside the bag.

While channel sampling probably provides the most accurate sample, the process is laborious, time-consuming, and expensive. In many cases satisfactory results may be obtained by "chip channel" sampling. In this method, the samples are laid out as for channel sampling, but instead of cutting out a channel, a band about 0.3 m (1 ft) wide is chipped with a geology pick, or hammer and chisel, across the width of the vein. An effort should be made to keep a relatively constant sample volume proportional to the widths of the vein. Care must be taken to collect approximately the same size chips across the zone being sampled. Chip points should also be as regularly spaced as possible.

In some cases it will be necessary to reduce this volume of the sample collected prior to sending it out for assay. It is common practice to make the first reduction at the mine, usually by the traditional method of "cone and quarter." However, the average fragment size of a sample should be reduced prior to any splitting operation. Several empirical methods for determining minimum fragment size with respect to type of mineralization are reported in the literature (Gy, 1968; Cummins and Given, 1973). Table 1 illustrates the data from one of these methods (Cummins and Given, 1973).

Diamond Drill Sampling: In recent years, as a result of improvement in technology and equipment, diamond drilling has become increasingly important as a sampling method. During exploration, the core is usually brought in from the drill, measured, weighed, and either split by a mechanical splitter or sawn lengthwise with a diamond saw. Half of the core is sent out for assay and the remaining half logged by a geologist and stored for reference. During later stages of exploration and development, it has been the practice at some properties to ship the entire core for assay, keeping only representative specimens or color photographs for reference.

Samples are usually collected at a constant interval down the length of the core although samples may be taken at shorter intervals through highly mineralized areas or veins. Often parts of the core known to be barren, or without visible evidence of mineralization, are not split or assayed.

In any drilling program, areas are likely to be encountered where drilling is difficult and core recovery is poor. In these zones it is common practice to collect samples of the drilling fluid, or sludge. McKinstry (1948) discusses various methods of integrating core and sludge analyses.

Miscellaneous Sampling Techniques: Besides the sampling methods mentioned previously, there are various other techniques employed in specific situations. These methods include such procedures as random "grab" samples, such as sampling broken muck, and large bulk samples, usually collected in lots of a half ton or larger. These large samples are taken for metallurgical testing or to improve sample accuracy in deposits such as epithermal gold deposits, where ore mineral distribution is particularly erratic.

Rotary or churn drill cuttings are utilized for sampling many types of industrial mineral deposits as well as disseminated-type metal ores. This drilling method is rapid and relatively low cost. The chips and fine material produced may be collected over uniform intervals, split if necessary, and bagged for analysis. The

Table 1. Minimum Permissible Sample Weight for a Given Particle Size

Diam of Largest Pieces, In.*	Diam of Largest Pieces, Mesh	Very Low Grade or Very Uniform Ore, Lb*	Medium Ores, Lb*	Rich or Spot Ores, Lb*
4		4,800	35,556	
2		1,200	8,889	51,200
1		300	2,222	12,800
0.5		75	556	3,200
0.25		19	139	800
0.131	6	5.15	38.1	220
0.065	10	1.29	9.5	55
0.0328	20	0.322	2.37	13.76
0.0164	35	0.081	0.59	3.44
0.0082	65	0.020	0.15	0.86
0.0041	150	0.005	0.038	0.21

*Metric equivalents: 1 in. × 25.4 = mm; 1 lb × 0.453 592 4 = kg. After Cummins and Given, 1973.

sampling of uranium and thorium deposits by radiometric logging is a specialized technique and will not be discussed in this presentation.

Assaying: Assaying may be done by a commercial laboratory or by an in-house company lab. In any case, a certain percentage of the samples, usually a minimum of 10%, should be assigned a new sample number and resubmitted for a repeat analysis. This provides a check on the analytical precision of the laboratory. It is also recommended practice to send a percentage of the samples to a different laboratory for accuracy comparison. Should there be any doubt as to the accuracy of the particular laboratory used, a few standard samples, including a blank, should be submitted for analysis.

Statistical Analysis of Sample Data: When the assays have been received from the laboratory and the validity of the results has been satisfactorily established, it is often useful to make some simple statistical analyses of the data. Classical statistical techniques are based on two assumptions: that the samples are random and that the data have a normal distribution. The problem of obtaining random samples is somewhat difficult to analyze. Samples collected from an ore deposit are seldom statistically independent of one another. This is a factor which is used to advantage in the calculation of geostatistical ore reserves, a subject which will be discussed in detail in a later section.

Several statistical techniques are known to determine if sample data are random. One of the easiest to calculate is the sample volume-variance relationship (Hazen, 1967). If samples are available in several different volumes, for example, from drill holes of various diameters, then if the sample data are random, the following relationship should be true:

$$S_1V_1 = S_2V_2 \ldots SnVn$$

where S is sample variance for one discrete sample volume and V is sample volume.

In many types of deposits, the fact that sample data are not random is not of serious concern. The results of the common statistical calculations may still be utilized while keeping in mind the bias introduced by the lack of statistical independence among the samples.

A normal sample distribution is seldom fulfilled with geological samples, though the logarithms of the data often show a normal distribution. In order to examine the type of statistical distribution for a data set, a simple histogram of the sample data may be prepared as shown on Fig. 1. Fig. 1 also illustrates some of the sample distributions common to mineral deposit data.

If an examination of the sample frequency distribution as shown on a histogram indicates an approximate normal distribution, then the normal statistical parameters may be calculated for the data set. The mean (average) variance and standard deviation calculated for the sample data should provide a reasonable approximation of these parameters for the deposit or portion of the deposit which has been sampled.

In those cases where examination of the sample distribution indicates a lognormal distribution, then the log parameters, geometric mean, and standard deviation should be calculated. The geometric mean will differ from the arithmetic mean, and in many types of deposits, particularly epithermal precious metal deposits, hydrothermal tin deposits as well as others, the geometric mean provides a more realistic approximation of true average grade for the deposit.

The normal and lognormal distributions are two types commonly encountered in mineral deposit sampling. Various other distributions are known but are

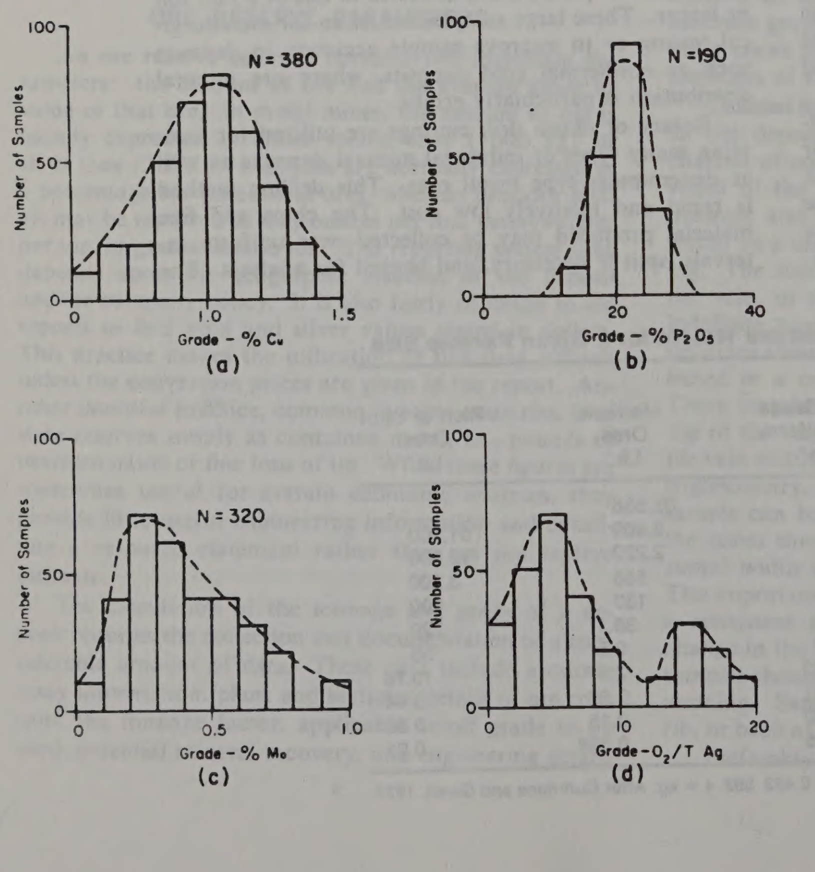


Fig. 1. Typical sample distribution. (a) Normal distribution, moderate variability, typical of some stratiform and massive sulfide deposits. (b) Normal distribution, low variability, found in certain industrial mineral, iron, and manganese deposits. (c) Lognormal distribution, common in many molybdenum, tin, tungsten, and precious metal deposits. (d) Bimodal distribution which may be produced by sampling two distinct ore types, or sampling across a zonation boundary in the mineralization.

beyond the scope of this discussion. For further information, see Hazen (1967) and Koch and Link (1970-1971).

If it can be assumed that the standard deviation of the samples provides a reasonable approximation of the standard deviation for the entire deposit population, then this figure may be used to establish a precision on the grade estimate. The standard deviation is used to establish the standard error of the mean, $S = s/\sqrt{N}$, where s is the sample standard deviation and N is the number of samples. In order to establish the confidence interval, the standard error of the mean (s) is multiplied by the statistical factor t, usually referred to as students t, a function which is based on desired confidence limits and the number of samples. The appropriate values of t for the desired confidence level may be found in any handbook of statistical tables. At the 95% confidence level and greater than 50 samples, the value for t is approximately 2.0. Thus, if a particular set of 60 samples has a mean of 8.5% lead with a standard deviation of 1.2, then the confidence interval for the mean would be:

$$CI = \frac{8.5 \pm 1.2 \times 2}{\sqrt{60}}$$
$$= 8.5 \pm 0.31$$
$$CI = 8.19 < 8.5 < 8.81$$

There is only one chance in 20 (5% probability) that the true grade lies outside the range of 8.19% to 8.81% Pb.

Alternatively, the formula for the computation of confidence limits can be rearranged such that:

$$N = \left(\frac{2 \times S}{\frac{CI}{2}} \right)^2$$

and the formula can then be used to approximate the number of samples that are used to establish a required precision for the estimate of the mean. For example, in the USBM definition of proven ore, the grade determination should be within 20% of the estimated value. If the calculation of a set of sample data indicates an average grade of 1.0% copper, then to be considered as proven, the grade must be between the confidence

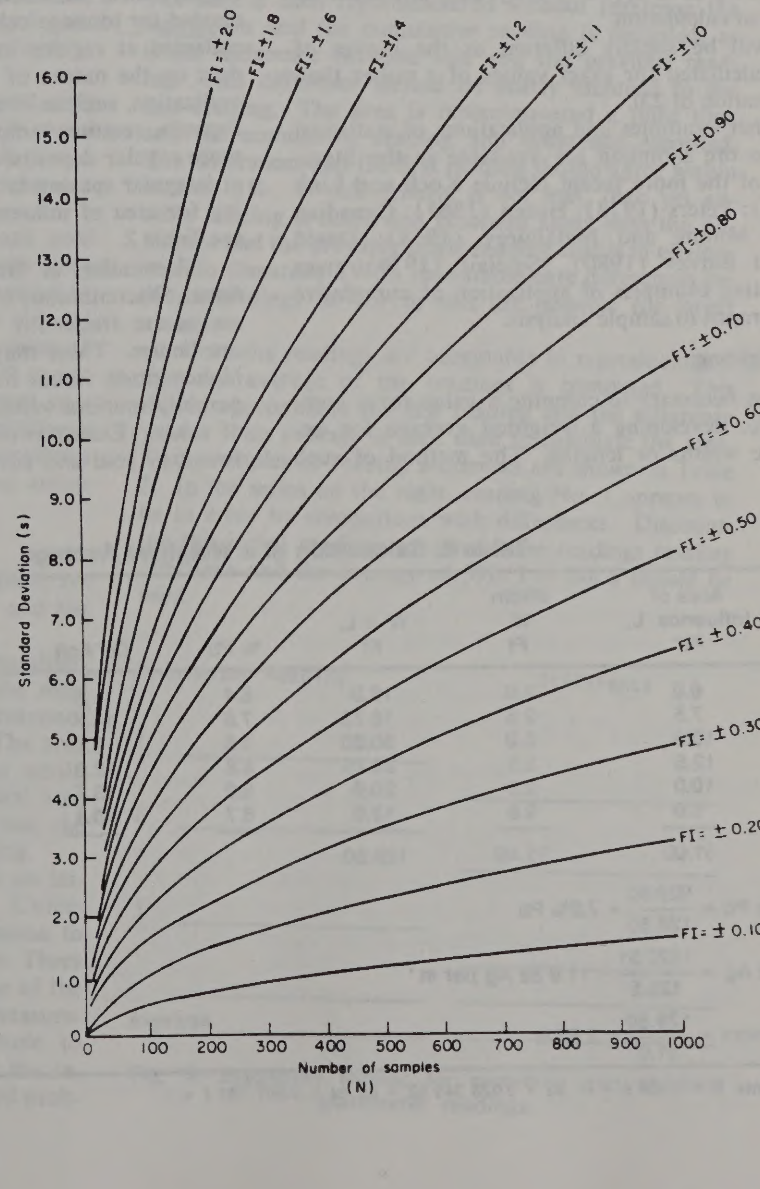


Fig. 2. Relationship between sample number, standard deviation, and confidence intervals at the 95% confidence level (after Hazen, 1961).

limits of 0.8% and 1.20% copper. If the standard deviation of preliminary samples is 1.5, then:

$$N = \left(\frac{2 \times 1.5}{\frac{0.4}{2}}\right)^2 = 225$$

If the preliminary sample consisted of 60 assays and produced a standard deviation of 1.5, then approximately 225 samples will be required before the grade of the deposit may be considered as meeting proven ore reserve standards. In actual practice, the standard deviation, as well as the error of the mean, will decrease as the sample number increases. Therefore, the confidence intervals should be recalculated as each group of samples comes in until the required precision is obtained. Peters (1978) illustrates an example of this same technique applied to a diamond drill program. Fig. 2, reproduced from Hazen (1961) is a graph of the previous equation for the 95% confidence level, and may be used for rapid approximations. For example, using Fig. 2 and the previous data where there is a standard deviation of 1.5 and confidence limits of ±0.2 are required, the expected number of samples is approximately 200, the same as for the original calculation.

Values will be slightly different as the curves of Fig. 2 are calculated for exact values of t rather than the approximation of 2.0.

Many other examples and applications of statistical techniques to ore sampling are available in the literature. Some of the more recent include Koch and Link (1970, 1974); Peters (1978); Hazen (1961); Canadian Institute of Mining and Metallurgy (1968); David (1973); and Barnes (1980). Sinclair (1976) gives some interesting examples of application of cumulative probability graphs to sample analysis.

Sample Weighting

It is often necessary to compute a value for a composite sample, developing a weighted average for unequal sample widths or lengths. The method of such weighted sample calculation is illustrated in Fig. 3, which shows a series of samples collected from a vein exposure in a raise in a hypothetical lead-silver mine. Note that the samples are collected at right angles to the dip of the bed and, in this case, the vein is assumed to extend at right angles to the plane of the page. If for some reason it is impossible to collect samples across the true width of the vein, the measured width should be corrected to the true width by a simple trigonometric calculation as the true width will be needed for tonnage calculations. Normally, samples are collected at regular intervals with the interval dependent on the nature of the mineralization. Erratic mineralization, such as is common in epithermal gold-silver deposits, requires sampling at much closer spacing than more regular deposits. Here, samples have been shown at irregular spacing to illustrate the principal of weighting for area of influence. For purposes of calculation, see Table 2.

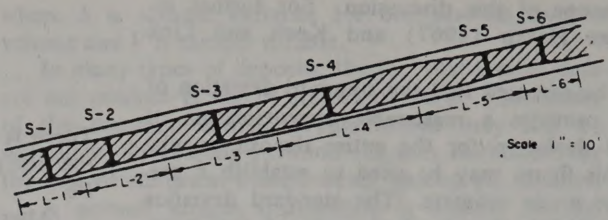


Fig. 3. Sample plan of a silver-lead vein. Metric equivalents: 1 in. × 25.4 = mm; 1 ft × 0.3048 = m.

Discounting of Irregular High Grade Assay Sections: Discontinuous or irregularly occurring high grade assays are frequently encountered in certain types of ore bodies. These may occur within a zone of slightly higher grade values or be encountered as a single apparently spurious value surrounded by the normal range of values. Examples of this phenomenon are commonly found in gold and silver deposits of all types, in vein-

Table 2. Calculation of a Weighted Average

Sample No.	Area of Influence, L, Ft*	Width, W, Ft	W × L, Ft	% Pb	Oz*Ag/t	Pb × W × L	Ag × W × L
S-1	6.0	3.0	18.0	6.4	11.3	115.20	203.40
S-2	7.5	2.5	18.75	7.6	14.7	142.50	275.63
S-3	10.0	3.0	30.00	5.6	8.6	168.00	258.00
S-4	12.5	2.3	28.75	8.8	12.9	253.00	370.88
S-5	10.0	2.0	20.0	8.2	13.7	164.00	274.00
S-6	5.0	2.6	13.0	6.7	10.8	87.10	140.40
Total	51.00	15.40	128.50			929.80	1522.31

$$\text{Average grade Pb} = \frac{929.80}{128.50} = 7.2\%\ \text{Pb}$$

$$\text{Average grade Ag} = \frac{1522.31}{128.5} = 11.9\ \text{oz Ag per st}^*$$

$$\text{Average thickness} = \frac{128.50}{51.0} = 2.52\ \text{ft}$$

*Metric equivalents: ft × 0.304 8 = m; oz × 0.028 349 52 = kg; st × 0.907 184 7 = t.

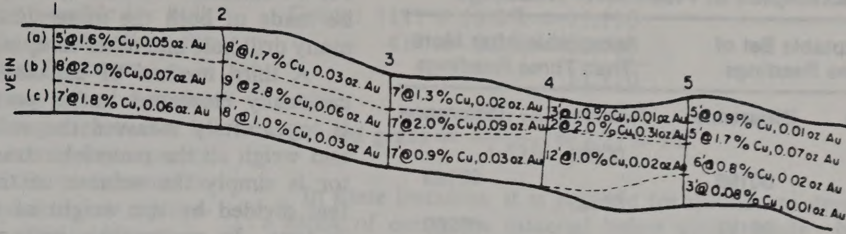


Fig. 4. Plan view of a gold vein. Metric equivalent: 1 ft × 0.3048 = m.

type uranium, tin and tungsten, as well as other deposits. Proper treatment of such anomalous high values is of particular importance in the high unit value commodities. Improper or imprudent use of anomalously higher grade assay samples for the high unit value commodities could have a seriously misleading impact on the reserve estimate and value of the deposit. As a general rule, if there is no correlatable zone of higher values, then the abnormally high assay should be reduced to the value of the highest adjacent assay. If during mining there is a greater value present than that anticipated, all well and good. If, however, as mining progresses, the grade of mineral encountered is considerably lower than has been indicated using nondiscounted anomalous high values in the ore reserve process, this could result in serious operating losses. A method of adjusting a series of samples for erratic high grade intervals is shown in Fig. 4.

In the next case, five crosscuts or horizontal drill holes cross a well-defined vein structure with three identifiable portions of the vein. This example could just as well be a vertical section through a tabular ore zone.

The 0.6-m (2-ft) sample of 8.6 g (0.31 oz) gold per ton in cut four is considerably higher than the samples surrounding it in cuts three and five or observed in samples in cuts one and two. To err on the side of caution, it should be discounted to either the value of the highest adjacent correlatable sample, i.e., 0.09 oz per ton in cut three or to the average of the adjacent correlatable samples in cuts three and five,.(0.09 + 0.07 oz) /2 = 0.08 oz per ton. Whichever of these two procedures is used should be used consistently during the entire reserve calculation procedure.

Tonnage Determination

The calculation of tonnage for an ore deposit requires that the volume of the mineralized zone and the tonnage conversion factor are known.

Volume Calculation: The volume of the mineralized zone is calculated by measuring the area of the mineralization and multiplying the area by the corresponding thickness of material above cutoff grade. The area may be estimated by breaking the area into small, regular geometric figures and calculating the total area by geometry. For very irregular or curved areas, the area is most easily determined by planimetering.

The measuring of areas by planimetering is an important part of many ore reserve estimations. Unfortunately, all too often insufficient care or attention to detail is given to this aspect of reserve estimation. There are several problem areas in planimetering. One of the major problems is that for smaller areas, small measurement errors can become of sufficient magnitude to seriously affect the area measurement and thus the reserve calculation. Another frequently encountered problem is an erroneous reading of the planimeter. Both these problems are essentially eliminated by the planimetering of all areas at least three times. A format that has proven exceptionally useful for recording and checking the accuracy (or reproducibility) of planimetering is shown in Fig. 5.

This format provides for continued cross-checking of the results and develops a record of the relative reliability of the readings. All readings are made consecutively without rezeroing the planimeter. This removes the initial random starting or missed zero error. Particular attention is given to zeroing the instrument at the start of planimetering of any one area. The area is then planimetered and the first reading recorded. The area is then replanimetered without rezeroing the instrument and the cumulative reading is recorded, as is the difference between this and the previous reading. This difference should be nearly identical to the first reading. The area is replanimetered a third time and the cumulative reading and difference recorded. The differences and the first reading should agree within at least 3%. If this criteria is not met, then yet another reading is taken and recorded along with the difference and the previous single reading that is in obvious error is discarded. With reasonable care differences between readings of 1% or less are possible for larger areas.

Once the readings are acceptable in reproducibility, then the average of the readings is computed. This should approximate the first reading and the difference values. This average is then used to calculate the area. Two examples of reading sequences are shown in Table 3. In the series on the right, reading No. 1 appears to be in error by comparison with differences. Discounting this initial reading, we have three readings totaling 596 units and the average of 596/3 = 198.6 should be used.

Planimeter Reading   Difference

1 ____________

____

2 ____________

____

3 ____________

____

____________ ____________

Average ____________ ____________

Fig. 5. Suggested format for recording and checking planimeter readings.

Table 3. Examples of Planimeter Readings

	Acceptable Set of Three Readings		Acceptable After More Than Three Readings	
	Readings	Difference	Readings	Difference
1.	00197		00208	
		00199		00199
2.	00396		00407	
		00197		00200
3.	00593	—	00607	00197
			00804	

Average: $\frac{593}{3} = 197.6$ (198)

Average discounting 1st Reading: $\frac{804-208}{3} = 198.6$ (199)

Tonnage Factor Calculation: The tonnage factor provides the mechanism for the conversion from volume of ore to weight of ore. In the English system, the tonnage factor is normally expressed as cubic feet per ton of ore. In the metric system, the tonnage factor is the specific gravity of the ore. The tonnage factor is dependent upon the specific gravity of the ore, and the specific gravity is a function of the mineral composition of the ore. Probably the most accurate method of determining specific gravity of an ore is to calculate an average specific gravity using specific gravities of individual minerals (Table 4), provided the relative percentages of ore minerals present are accurately known. For example, if a massive sulfide ore is 10% galena, 35% sphalerite, and 55% pyrite, the specific gravity would be:

$$
\begin{aligned}
7.6 \times 0.10 &= 0.76 \\
4.1 \times 0.35 &= 1.44 \\
5.0 \times 0.55 &= 2.75 \\
\hline
4.95 &= \text{sp gr of ore}
\end{aligned}
$$

The specific gravity of an ore may also be computed by weighing a core or specimen of the ore in air, then weighing the same sample suspended in water. The specific gravity is calculated by the following formula:

$$\text{Sp gr} = \frac{Wa}{Wa - Ww}$$

where Wa is weight in air and Ww = weight in water.

If the ore volume has been computed in cubic meters, the volume multiplied by the specific gravity is the tonnage in metric tons directly. If working in the English system, the tonnage factor is calculated as follows:

$$\text{Sp gr} \times 62.5 \text{(lb per cu ft water)} = \text{lb per cu ft ore}$$

$$\text{Tonnage factor} = \frac{2000 \text{ lb per ton}}{\text{lb per cu ft ore}}$$

$$\text{Tonnage factor} = \text{cu ft per ton}$$

For example, if a porphyry copper ore has a specific gravity of 2.8, then:

$$2.8 \times 62.5 \text{ (lb per cu ft of water)} = 175 \text{ lb per cu ft ore}$$

$$\text{Tonnage factor} = \frac{2000}{175} = 11.43 \text{ cu ft per ton ore}$$

For purposes of ore reserve estimation a single, or even a few, samples of core or ore specimens, would not be suitable. Specific gravity determinations would be made of both the mineralization and gangue from many drill hole and other samples.

A third method of calculation, which is only occasionally employed but is preferred by the authors, is to carefully measure the volume of an excavation and weigh all the material extracted. The tonnage factor is simply the volume of the excavation in cubic feet divided by the weight of the material recovered in tons. In reality, this bulk density procedure pro-

Table 4. Specific Gravity of Common Rocks and Minerals

	Specific Gravity
Rocks	
Andesite	2.4–2.8
Basalt	2.7–3.2
Diabase	2.8–3.1
Dolomite	2.7–2.8
Gabbro	2.9–3.1
Granite	2.6–2.7
Gravel (dry)	1.6–2.0
Limestone	2.7–2.8
Rhyolite	2.2–2.7
Sandstone	2.0–3.2
Shale	1.6–3.0
Schist	2.6–3.0
Minerals	
Anhydrite	2.9
Anglesite	6.3
Argentite	7.3
Arsenopyrite	6.0
Barite	4.5
Bauxite	4.5
Bornite	4.9
Calcite	2.7
Cassiterite	7.0
Cerussite	6.5
Chalcedony	2.6
Chalcocite	5.7
Chalcopyrite	4.3
Chromite	4.5
Copper	8.8
Covellite	4.6
Cuprite	6.0
Feldspar	2.6–2.8
Fluorite	3.1
Galena	7.6
Gold	17.5
Graphite	2.2
Gypsum	2.3
Hematite	5.2
Molybdenite	4.8
Muscovite	2.9
Pentlandite	4.8
Platinum	19.0
Pyroxene	3.3
Pyrite	5.0
Pyrrhotite	4.7
Quartz	2.7
Scheelite	6.0
Sericite	2.6
Silver	10.6
Smithsonite	4.4
Sphalerite	4.1
Stibnite	4.6
Sulfur	2.1
Uraninite	9.4

vides the most accurate tonnage factor possible for bulk material sampled. Other procedures do not allow for differences related to void spaces such as fractures.

Problems are encountered with use of a constant tonnage factor where the deposit changes character in different portions of the body. In such cases, separate tonnage factors for each portion of the body must be calculated. For massive sulfide deposits and replacement deposits with simple mineralogy it is often possible to prepare a nomograph relating tonnage factor to assay data. The tonnage factor used for ore is keyed to changes in the ore content and grade.

Engineering Considerations

Before proceeding with an explanation of various methods of reserve computation, a brief discussion of pertinent engineering factors is in order.

Geological Considerations: In many instances, particularly in the exploration stage of a project, it is common practice to project ore extensions based on geologic inference. These projections should never be extended across geological discontinuities such as faults, contacts, unconformities, fold axes, etc. until positive ore correlation data is available on both sides of the discontinuity. Preliminary drilling and other sampling will give an indication of the nature of ore boundaries, whether they are sharply defined or gradational. Lateral or vertical mineralogical zonation, development of discrete ore shoots, and other potential problems will become apparent as exploration work progresses.

Mining and Metallurgical Recovery: Once the deposit is reasonably well-defined as to its limits, shape, and character, consideration can be given to selection of an appropriate mining method, and then an estimate can be made concerning percentage extraction from the deposit. The portion of mineralization above the cutoff value that can actually be exploited constitutes the *minable ore reserve.*

Metallurgical recovery may be a critical factor in the economic analysis of a property. However, it usually is used only in calculation of a cutoff grade during reserve computation for a deposit. Any increase or decrease in metallurgical recovery has the same effect as an increase or decrease in grade of ore mined.

Dilution: Dilution is the unavoidable extraction of barren or below cutoff grade material along with the ore. In vein deposits the most common source of dilution is blasting overbreak in the walls of the deposit. Dilution may be handled in various ways. In veins which have at least one gradational or "assay" wall, it is common to cut samples over the normal mining width and use an average grade and tonnage factor for the entire interval. In the case of narrow veins with sharp boundaries, when wall rock must be taken with the vein, the dilution may be calculated as follows:

Ore block: 30.48 × 15.25 m (100 × 50 ft)
Average width: 0.6 m (2.0 ft)
Average grade: 10.0% Pb
Tonnage factor: 9.0 cu ft per ton ore
12.0 cu ft per ton wall rock
Minimum mining width: 0.9 m (3 ft)
Ore tons: $100 \times 50 \times 2.0 \div 9 = 1111$ tons
Waste tons: $100 \times 50 \times 1.0 \div 12 = 417$ tons
Total tons mined = 1528 tons

$$\text{Grade: } \begin{array}{r} 1111 \times 10.0\% = 11{,}110 \\ 417 \times 0.0\% = 00{,}000 \\ \hline 11{,}110 \end{array}$$

$$\text{Diluted grade} = \frac{11{,}110}{1{,}528} = 7.27\% \text{ Pb}$$

In some instances, it is possible for dilution to drop a block of ore grade material below cutoff grade. In such instances, the block economically ceases to be ore and must be left until the cutoff grade is lowered, or greater selectivity in mining can be made. One common practice with thin veins is to "double shoot" the vein—the vein is drilled, blasted, and removed, then the wall rock is blasted for fill. However, in such cases the mineral value in the ore mined and milled must cover the cost of blasting the waste material.

As mentioned previously, there are various methods of accounting for dilution in mining. One technique which is not permissible should perhaps be mentioned. One of the authors once observed a case in which the average ore grade of a tin deposit was being upgraded when the vein width exceeded the minimum mining width. This is not acceptable engineering practice.

Cutoff Grade: As stated before, the cutoff grade is the minimum grade that can be mined at a profit. As economic conditions change, the cutoff grade may increase or decrease. It is common practice to compute the ore reserves of a mine for various cutoff grades and plot the results as a series of grade-tonnage curves. These curves should be updated regularly to aid in mine planning.

METHODS OF CALCULATION

In this section, several of the common traditional reserve calculation methods are explained and illustrated by simple examples. The methods discussed are calculation by mining block, calculation by polygons, calculation by triangles, and calculation by section. There are many other calculation methods which have been presented in the literature, although many are only somewhat sophisticated variations of the methods mentioned previously. No one technique is universally applicable to all deposits. Methods such as mining blocks and sections work well in steeply dipping veins and tabular deposits, whereas polygonal methods have found wide application to disseminated and flat-lying bedded deposits. The method selected for any particular deposit will be dependent upon the geological and engineering elements unique to each deposit and usually the ore reserves will be calculated several different ways as an internal audit.

While simple computer programs may be written to handle any of the following calculation methods, all that is really required is the basic data, maps, sections, a hand calculator or slide rule, and considerable patience. More sophisticated computer-oriented methods such as interpolation using inverse distance weighting functions are available but not discussed in this presentation. A review of geostatistical methods is provided as an appendix.

Calculation by Mining Block

Fig. 6 shows the method of estimating the tonnage and grade of an ore block in a vein type mine. Samples

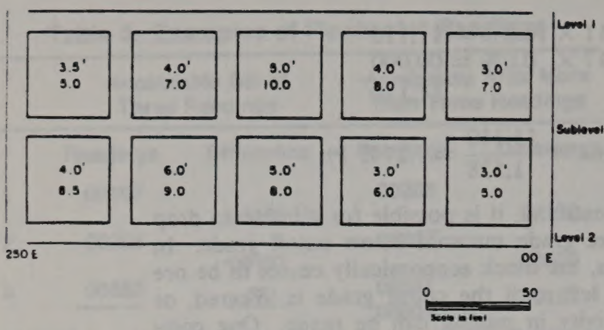


Fig. 6. Longitudinal projection of an ore block. Metric equivalent: 1 ft × 0.3048 = m.

have been assumed to be cut at regular intervals. The vein in this area is assumed to exceed the minimum mining width of 0.91 m (3 ft). The method of calculation is simply an extension of the method of sample weighting illustrated in Fig. 3.

Fig. 6 represents a block of ore between two levels 30.5 m (100 ft) apart, and coordinates 00E and 250E. The upper figure in the block is the average thickness and the lower figure is the average grade in ounces of silver per ton. The tonnage factor is 9 cu ft per ton. The average grade and tonnage for the block is computed in Table 5.

Calculation by Polygons

The method of calculation by polygons is often used with drill-hole data. Polygons may be constructed on plans, cross sections, or longitudinal sections. The polygons, once constructed and ranked as to class of ore, are planimetered to determine the area of mineralization. The thickness of above cutoff grade mineralization is applied to the entire polygon to establish the volume estimate. In this method, the average grade of mineralization encountered by the sample point within the polygon is considered to accurately represent the grade of the entire volume of material within the polygon.

The construction of polygons is quite simple. The method assumes that the area of influence of any sample point extends halfway to the adjacent sample points. The procedure for construction of polygons is illustrated in Fig. 7. It shows five drill holes (a), connecting lines for the drill holes (b), the construction of perpendicular bisectors of the lines between adjacent drill holes (c), and the final polygon (d).

The polygon method makes the basic assumption that the area of influence of a drill hole extends halfway to the next adjacent hole. An alternative viewpoint is that of a circular area of influence for drill-hole intercepts. The concept of circular area of influence about a mineralized intercept can be used to assign the relative classification to the polygon blocks. One important aspect of the circular area of influence for

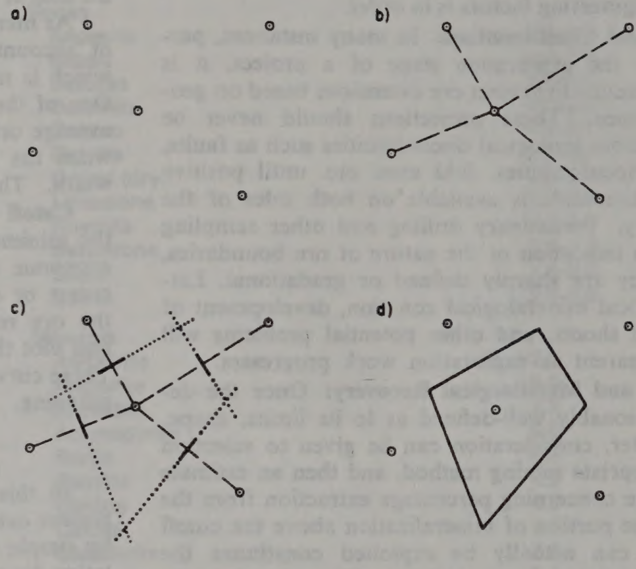


Fig. 7. Construction of polygons. (a) Drill-hole plan. (b) Connecting lines between drill holes. (c) Construction of perpendicular bisectors of connecting lines. (d) Construction of final polygon.

Table 5. Calculation of Ore Blocks

Length		Width		Thickness	Tonnage Factor		Tons*		Grade		Tons-Grade
50	×	50	×	3.5	9	=	972	×	5	=	4,861
50	×	50	×	4.0	9	=	1,111	×	7	=	7,777
50	×	50	×	5.0	9	=	1,388	×	10	=	13,889
50	×	50	×	4.0	9	=	1,111	×	8	=	8,888
50	×	50	×	3.0	9	=	833	×	7	=	5,833
50	×	50	×	3.0	9	=	833	×	5	=	4,166
50	×	50	×	3.0	9	=	833	×	6	=	5,000
50	×	50	×	5	9	=	1,388	×	8	=	11,111
50	×	50	×	6	9	=	1,666	×	9	=	15,000
50	×	50	×	4	9	=	1,111	×	8.5	=	9,444
							11,246				85,972

Total tons = 11,246

$$\text{Average grade} = \frac{85972}{11246} = 7.64 \text{ oz* Ag/ton}$$

*Metric equivalents: 1 oz × 0.028 349 52 = kg; 1 st × 0.907 184 7 = t.

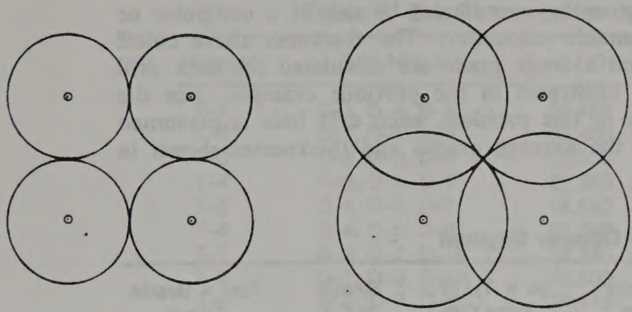


Fig. 8. Polygon classification by circular area of influence. Left, incorrect method; right, correct method.

drill holes is shown in Fig. 8. This is completely covering the area contained within a square grid of holes by the circular area of influence of the holes requiring a drill-hole spacing of $(r)(\sqrt{2})$, where (r) is the radius of influence. This means that although a drill-hole spacing of 60.96 m (200 ft) in a porphyry copper deposit would more than reasonably define proven ore grade mineralization, the radius of the circle of influence for that spacing is in reality $(200)(\sqrt{2}) = 86.2$ m (282.8 ft).

Once the area of influence of the drill holes corresponding to proven and probable ore has been determined, it can be used to check and rank the polygons. A rapid, easy, and acceptable method of ranking polygons is to draw a set of circles that represent the radius of influence for proven, probable, and possible ore reserve categories on a stable base material and then overlay the finished polygon map over this on a light table. Each drill-hole polygon is superimposed on the center of the "proven" area of influence circle matching the center points. If the polygon falls completely within the "proven" range circle, the polygon is marked as belonging to the proven category. Subsequent checks are made of the polygons not meeting the criteria to be classified as proven blocks using the other area of influence circles until all the polygons are ranked. For record and bookkeeping purposes, polygons are conveniently referenced to the drill-hole number and the section or level being evaluated, e.g., polygon DDH-8-16, level 2080. Fig. 9 illustrates the computation of an ore block by the polygon method.

The method of calculation by polygons is often used with drill sample data. The method makes the assumption that the area of influence of each drill hole extends half the distance to each adjacent drill hole, with appropriate modifications for known geologic factors such as faults, contacts, or mineralization limits. The areas of the polygons may be measured by planimeter or calculated geometrically by breaking up each polygon into a series of triangles. The average grade and thickness of each drill hole may be determined as shown in Table 6 for drill hole D-1. In this example, the mineralization is assumed to be copper, and the cutoff grade is 0.40% Cu.

The average grade and thickness is determined for each drill hole, and the reserves are calculated as shown on Table 7. Each polygon is labeled for the contained drill hole.

Computation by Triangles

Another method of computing reserves is a modification of the polygon method. In this method a series of triangles is constructed with the drill holes at the apices. This method has the advantage in that the three points are considered in the calculation of the thickness and grade parameters for each triangular

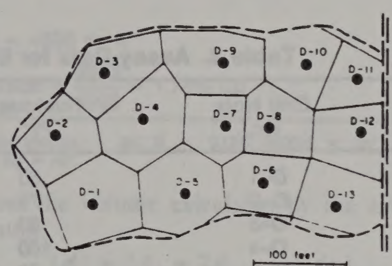


Fig. 9. Diamond drill-hole plan of the Bonanza copper deposit. Metric equivalent: 1 ft × 0.3048 = m.

Table 6. Assay Data for Drill Hole D–1

Interval, ft*	Thickness, ft	Grade, %Cu	Grade x Thickness	
0–100	100	0.31	0.00	(Below cutoff)
100–110	10	0.47	4.70	
110–122	12	0.73	8.75	
122–130	8	0.96	7.68	
130–150	20	1.04	20.80	
150–200	50	0.82	41.00	
200–220	20	0.54	10.80	
220–250	30	0.42	12.60	
250–270	20	0.35	0.00	(Below cutoff)
	150		106.43	Thickness and grade-thickness above cutoff

$$\text{Average grade} = \frac{106.43}{150.00} = 0.71\%\ \text{Cu}$$

Thickness = 150 ft

*Metric equivalent: 1 ft × 0.3048 = m.

reserve block. The construction and calculation of ore reserves by use of triangles are shown in Fig. 10.

The method of calculation by triangles is a modification of the polygonal method in which the drill area is divided into triangles by connecting adjacent drill holes with construction lines. This method has the advantage that the areas are easily calculated by geometry or by coordinates by use of a computer or programmable calculator. The thickness above cutoff grade and average grade are calculated for each drill hole, as illustrated in the previous example. For the purposes of this problem, each drill hole is presumed to have the average grades and thicknesses shown in Table 8.

Table 7. Ore Reserves for Bonanza Copper Deposit

Polygon	Area-A Sq Ft	Thickness, T, Ft	A × T, Cu Ft	Tonnage Factor, Cu Ft*/Ton	(A × T)/TF, Tons Ore	Grade % Cu	Ton × Grade, Ton %
D-1	5,320	150	798,000	12.5	63,840	0.71	45,326
D-2	5,300	135	715,500	12.5	57,240	0.66	37,778
D-3	4,400	180	792,000	12.5	63,360	0.82	51,955
D-4	5,520	175	966,000	12.5	77,280	0.75	57,960
D-5	6,800	155	105,400	12.5	84,320	1.00	84,320
D-6	4,960	180	892,800	12.5	71,424	0.97	69,281
D-7	4,520	250	1,130,000	12.5	90,400	1.21	109,384
D-8	4,640	240	1,113,600	12.5	89,088	1.36	121,159
D-9	5,840	150	876,000	12.5	70,080	0.93	65,174
D-10	4,840	135	653,400	12.5	52,272	0.87	45,476
D-11	3,760	120	451,200	12.5	36,096	0.81	29,237
D-12	4,270	165	637,200	12.5	50,976	0.75	38,232
D-13	4,800	135	648,800	12.5	51,840	0.68	35,251
					858,216		790,553

Tons ore 858,216
Average grade 0.92% Cu

*Metric equivalents: 1 ft × 0.304 8 = m; 1 sq ft × 0.092 903 04 = m²; 1 cu ft × 0.028 316 85 = m³; 1 st × 0.907 184 7 = t.

Table 8. Assay Data for the Ojala Copper Deposit

Drill Hole No.	Thickness Ft*	Average Grade, % Cu
D-1	50	0.93
D-2	75	0.77
D-3	60	0.82
D-4	100	1.05
D-5	75	0.72
D-6	60	0.49
D-7	105	1.63
D-8	80	0.91
D-9	70	0.86
D-10	75	0.74

Given this data, the tonnage and grade calculation for Triangle T-1 would be as follows.

Area = 4400 sq ft* (by geometry)

Average Grade-Thickness for Triangle T-1

Drill Hole	Thickness, ft	Average Grade, % Cu	Grade × Thickness ft %
D-1	50	0.93	46.50
D-4	100	1.05	105.00
D-5	75	0.72	54.00
	225		205.50

$$\text{Average grade} = \frac{205.50}{225.00} = 0.91\%\ \text{Cu}$$

Tonnage = area × average thickness × tonnage factor

$$= 4400 \times \frac{225}{3} \times \frac{1}{12.5} = 26{,}400\ \text{st}^*$$

*Metric equivalents: 1 ft × 0.3048 = m; 1 sq ft × 0.092 903 04 = m²; 1 st × 0.907 184 7 = t.

Table 9. Ore Reserves for the Ojala Copper Deposit

Triangle	Drill Holes	Tons* Ore	Average Grade	Tons × Grade
T-1	D-1, D-4, D-5	26,400	0.91	24,024
T-2	D-1, D-2, D-4	26,400	0.94	24,816
T-3	D-2, D-3, D-4	22,500	0.91	20,475
T-4	D-3, D-4, D-7	22,260	1.23	27,380
T-5	D-4, D-6, D-7	18,550	1.15	21,332
T-6	D-4, D-5, D-6	27,260	0.79	21,535
T-7	D-6, D-7, D-10	26,240	1.07	28,076
T-8	D-7, D-9, D-10	40,500	1.15	46,575
T-9	D-3, D-7, D-8	24,418	1.20	29,301
T-10	D-7, D-8, D-9	28,917		34,411
		263,445		277,927

Tonnage = 263,445 st

$$\text{Average grade} = \frac{277{,}927}{263{,}445} = 1.05\%\ \text{Cu}$$

*Metric equivalent: 1 st × 0.907 184 7 = t.

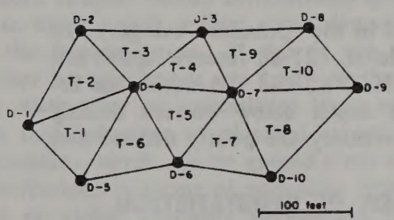


Fig. 10. Diamond drill-hole plan of the Ojala copper deposit. Metric equivalent: 1 ft × 0.3048 = m.

In like manner, the tonnage and grade for each triangle can be computed and Table 9 constructed.

Calculation by Section

The basis of this method is to calculate a block of ore that is bounded by regularly spaced cross sections (see Tables 10 and 11). The following equation illustrates the detailed calculation of a typical block of ore by the cross section method. The ore outline of each bounding section is divided into areas of influence based on the drill hole or other sample data. The areas of influence are then either planimetered or calculated geometrically. The individual areas are totaled for each section and the volume calculated by the average and area formula:

$$V = \frac{(A_1 + 2A_2 + 2A_3 \ . \ . \ . \ An)}{2} \times L$$

where A is area on section and L is a constant section spacing, or when using only two adjacent sections:

$$V = \frac{(A_1) + (A_2)}{2} \times L$$

Table 10. Assay Data for Section 100N, Big Rat Copper Vein

Sample No.	Area of Influence	Grade, % Cu	% Cu × sq ft*
T-1	A = 510 sq ft*	0.80	0408
DDH-1	B = 1000	2.55	2550
DDH-2	C = 1040	1.66	1726
C-1N	D = 710	1.70	1207
	3260		5891

Total Area 3260

$$\text{Average grade} = \frac{5891}{3260} = 1.81\%\ \text{Cu}$$

*Metric equivalents: sq ft x 0.092 903 04 = m²; cu ft x 0.028 316 85 = m³.

Table 11. Assay Data for Section 200N, Big Rat Copper Vein

Sample No.	Area of Influence, sq ft	Grade, % Cu	% Cu × sq ft*
T-2	A′ = 848	0.92	780
DDH-4	B′ = 1792	2.32	4157
DDH-3	C′ = 1280	1.59	2035
C-2N	D′ = 976	1.63	1591
	4896		8563

Total area = 4896 sq ft*

$$\text{Average grade} = \frac{8563}{4896} = 1.75\%\ \text{Cu}$$

*Metric equivalents: 1 sq ft × 0.092 90304 = m²; 1 cu ft × 0.028 316 85 = m³.

The volume is then converted to tons by application of the appropriate tonnage factor.

Fig. 11 shows two cross sections spaced 30.48 m (100 ft) apart. These sections show a tabular dipping vein sampled by a surface trench, two drill holes per section, and one crosscut per section. The vein is assumed to be copper ore with a tonnage factor of 9.5 cu ft per ton.

Block Average Grade:

Section	Area (sq ft)	Average Grade, % Cu	% Cu Ft²
100*N*	3260	1.81	5901
200*N*	4896	1.75	8563
	8156		14463

$$\text{Average block grade} = \frac{14463}{8156} = 1.77\%\ \text{Cu.}$$

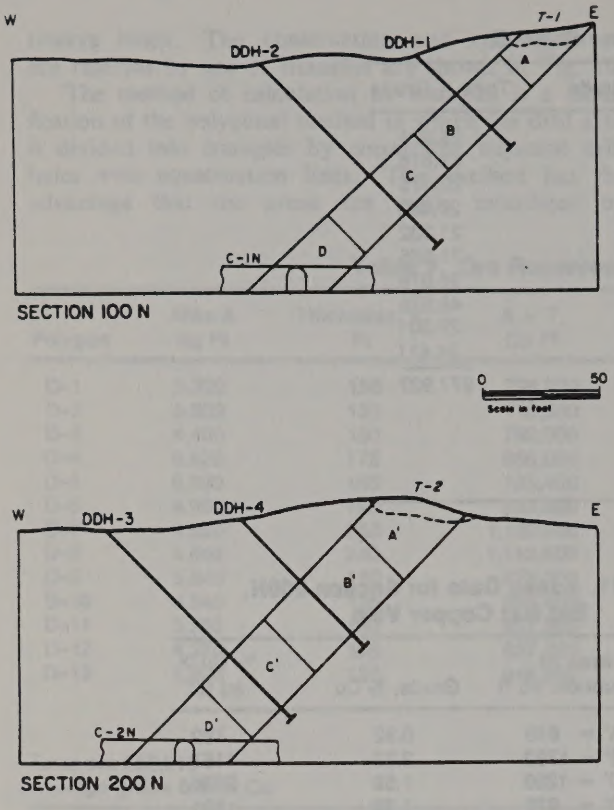


Fig. 11. Cross sections of the Big Rat copper vein. Metric equivalent: 1 ft × 0.3048 = m.

Volume of Ore Block:

$$v = \frac{\left(\begin{array}{c}\text{area section } 100N + \text{area section } \times \\ 200N \times \text{section spacing}\end{array}\right)}{2}$$

$$v = \frac{3260 + 4896 \times 100}{2} = 407{,}800 \text{ cu ft}$$

$$\text{Tonnage} = \frac{407800}{9.5} = 42{,}926 \text{ st}$$

The geologic reserve of this deposit between 100*N* and 200*N* is 38 934 t (42,926 st) with an average grade of 1.77% Cu. Similarly, the reserve calculations can be extended north and south to cover the entire minable strike length of the vein by adjacent pairs of sections.

Documentation

It is extremely important, no matter what ore reserve estimation methodology is used, to carefully document the method used and the limits applied. If the method is documented, anyone using the reserve figures will have a fuller understanding of the confidence limits inherent in the stated reserves. Also, in case of any change of mine personnel, it will be possible for future calculations to be consistent with the previous reserves data. Adherence to this policy also allows for future critical review of the applicability of the methodology to the specific deposit. Should a problem be encountered with the application of the reserve estimation method to the deposit, it may be possible to correct the problem without completely recalculating the entire reserve inventory.

The writing-up of the reserve estimation methodology also forces the staff to fully identify potential problem areas and to define methods of handling such problems in a consistent manner. The documentation should be prepared in the form of a manual keyed to the specific property by examples. Adequate space should be provided for notes to be added by the staff as problems or questions arise or ideas for improving the reserve calculation method are encountered. Such an ore reserve preparation manual provides each new staff member with a set of uniform procedures and guidelines to be used on the specific property.

The cutoff limits used, both those related to mining method and grade should be specified in the reserves statements. Examples of such qualifying statements include:

> proven 3,795,000 tons averaging 2.15% Cu
> probable 5,600,000 tons averaging 2.69% Cu
> possible 2,901,000 tons averaging 2.80% Cu
> based on a minimum mining width of 3.04 m (10 ft) of material above a cutoff grade of 0.80% Cu.

Note (1) all reserves in the "possible" category are located below the 3500 level, (2) all reserve figures are reported to the nearest 1000 st, and (3) additional inferred reserves exist at depth below current mining limits. These reserves presently are poorly defined and are hence not reported.

APPENDIX: REVIEW OF GEOSTATISTICAL METHODOLOGY

Introduction

Geostatistics as an ore reserve estimation methodology emerged in France in the early 1960s from the work of Georges Matheron and was based on original studies by Danie Krige in South Africa. The techniques are not merely an amalgamation of the geological sciences with probability theory and classical statistics, but rather an entirely new methodology.

It is not possible within this discussion to give a comprehensive treatment of the theory and application of geostatistics. The objectives here are merely to develop an overview of this relatively new methodology, to elaborate on some of its virtues and problems, and to provide some preliminary understanding of the basic techniques and nomenclature.

The primary purpose of any natural resource estimation method is to reliably estimate the overall ore reserves and the distribution of in situ and recoverable tonnages and grades throughout the deposit. Conventional methods (i.e., area of influence or polygonal, other geometrical, distance weighting, etc.) may provide a good total or global estimate of an ore body's reserves. However, a good geostatistical reserve study with careful attention to geologic controls on mineralization will provide not only a good total reserve estimate, but also a more reliable block by block reserve inventory with an indication of relative confidence in the block grades estimated.

Obviously, geostatistical methods, like any others, cannot increase the quantity of basic sample information available, nor can they improve the quality or accuracy of the basic assays. However, geostatistics properly applied does derive from the raw data the best possible estimates of ore-body parameters. This is par-

ticularly critical with marginal grade deposits, where a relatively minor change in anticipated mill head grade can have quite dramatic effects on the project's profitability. Geostatistical techniques should be regarded as a comprehensive suite of ore reserve estimation tools which, if they are appropriately understood and used, should generally lead to few surprises when the mine comes into production.

Classical statistics and tests have been used in ore reserve evaluation for many years. These methods do assume, however, that samples taken from an unknown population are randomly selected and are independent of each other. In the context of an ore body, this implies that the position from which any sample was taken is not important. Theoretically, using classical statistics, taking samples on opposite sides of an ore body would be just as good as taking them a short distance apart. Sample assays taken from holes drilled in close proximity to one another, within an ore body, obviously should not be random or independent. Closely spaced samples should demonstrate some correlation or, in other words, reflect some degree of continuity in the mineralization. If this is not the case, there is either no continuous ore body or the samples have been taken over an excessively large spacing.

Despite the limitations of classical statistics in ore reserve estimation, much can be gained from studies of sample distributions in terms of providing estimates of the overall or global ore-body parameters (ie., total ore tonnage, grade, and metal content). Serious mistakes, however, can occur if this theory is applied on a small scale. Even in the global case, one must exercise considerable caution in making predictions from sample grade distributions of recoverable ore tonnages and grades above a particular cutoff grade. This is necessary primarily because drilling pattern irregularities are very common and hence the grade distribution may be biased (high grade areas are typically drilled out more thoroughly than low grade areas). Secondly, the variance or dispersion of basic sample assay values is much greater than the variance of large blocks which can be selectively mined as ore or waste.

Unlike classical statistical approaches, geostatistics recognizes that samples in an ore deposit should be spatially correlated with one another, and that nearby samples will probably not be independent. The techniques are based on Matheron's idea of "regionalized" variables or, in other words, variables which are associated both with a volume (called a "support" in geostatistics) and a position in space. Thus, geostatistical methods utilize an understanding of the interrelationship of assay data within various geologic ore control regimes of a deposit. Geostatistics thus represents a major advance in ore reserve estimation technology.

The relationship between sample variance and size (decreasing variance with increasing size of the support) is utilized in developing estimates of recoverable ore reserves given a certain degree of mining selectivity and given a cutoff grade. The amount of spatial correlation or continuity is determined by the primary geostatistical tool—the variogram.

Variograms, representative of the mineralization's characteristics, are a prerequisite to any geostatistical ore reserve estimation. If the sampling density is too low for any underlying correlation to be detected, if the ore body is extremely homogeneous, or if poor sample collection, preparation, and assaying procedures were used, then no structure or continuity will be visible in the variogram. In this case, geostatistical methods become those of classical statistics, where samples can be assumed independent and where the best estimate of any block of ground is the average grade of all samples within the entire deposit. Under these circumstances, reliably predicting which part of the deposit is of ore grade and which is waste is not possible with any acceptable level of confidence.

Geostatistics theory utilizes somewhat foreign terminology and relatively advanced statistical concepts involving Lagrange multipliers, triple integration, etc. Only recently have case studies which have clearly indicated the potential benefits in applying geostatistical techniques appeared in the literature. In addition, early publications in this field were in French, and even those in English were not readily comprehensible to the average geologist or mining engineer. Because of these factors, geostatistics has sometimes been regarded skeptically as a method suited only to the biggest computer manned by systems analysts, who probably have little real appreciation for the geology or mining implications of the ore body which they are studying.

While the "black box" computer approach to geostatistics has been historically true at some operations, it should not, and need not, exist. Certainly the methodology does entail a relatively large amount of computation, somewhat "heavy" mathematical and statistical terminology, and generally does require the use of a computer. This, however, does not mean that the work need be done in isolation from the geologic staff. In fact, geologic cross sections, bench plans, and most importantly, the acquired understanding of the ore body by the geologic staff in terms of the lithologic, structural, or other controls on the mineralization, is of paramount importance in any geostatistical study.

A geostatistical ore reserve study will generally entail the following steps:

Study of the potential geologic controls on the grade of mineralization, and any zoning of the deposit through geologic interpretation, with the aid of statistical analysis correlating grade with host rock type, alteration type, fracture density, etc.

Computation of variograms within each geologic zone, interpretation of any differences in the continuity of mineralization in different directions (anisotropy), and selection of suitable variogram models for use in kriging.

Division of the ore body into three-dimensional matrices of blocks, panels, or cells and estimation of the in situ grade and the estimation error for each block from the surrounding sample values using kriging (in situ or geologic reserves). Using properly selected limits on block estimation errors, each block can be subsequently classified into the standard "measured," "indicated," and "inferred" categories.

Study of the distribution of sample values, selection of suitable models, and estimation of the tonnage proportion of each in situ block and its average grade which can be recovered above a given cutoff grade with a given degree of mining selectivity (recoverable reserves at a particular cutoff grade).

Printing of recoverable grade distribution plans on

a convenient scale by each level or bench for mine planning purposes.

The Variogram

Computation of variograms within like geologic zones is the first step in any geostatistical ore reserve study. Variograms are used in all subsequent phases, including kriging. Even if kriging is not undertaken, a variogram study is of invaluable assistance in quantitatively defining the traditional concept of "area of influence" of a sample within a particular deposit. Moreover, it can be used in determining the optimum drill-hole spacing to define the reserves at a particular level of confidence.

A variogram, if properly determined within similar geological zones, numerically describes the way in which an ore-body parameter (i.e., grade, thickness, accumulation, bedrock elevation, etc.) is spatially correlated. It expresses the similarity or dissimilarity between the parameter values as it changes with distance.

The mathematical formulation of a variogram function is as follows:

$$\vec{\gamma}(h) = \frac{1}{2n}\sum_{i=1}^{n}\left[Z(x_i) - Z(x_i + h)\right]^2$$

where $Z(x_i)$ is the value of the regionalized variable (e.g., grade) at point x_i, $Z(x_i + h)$ is the grade at another point at a distance (h) from the point, x_i, and n is the number of sample pairs.

The sample pairs are each oriented in the same direction [hence the vector notation $\gamma(h)$], are each separated by the same distance (h) m, and are of equivolume (i.e., constant variogram support).

In practice, the squared differences between all pairs of sample grades at a distance (h) apart are averaged. The process is repeated for different values of the distance (h), and the half averaged squared differences in grade, called $\gamma(h)$, are then plotted against the corresponding distance (h). The resulting graph is often known as an experimental variogram, because it is based only on samples, and is aiming at representing the true underlying variogram of the deposit.

Often there is evidence of a relationship between mean grade and variance, i.e., high variances associated with high grade material which is known as "proportional effect" in geostatistics. If this is found to be the case, a so-called "relative" variogram should be computed. This involves dividing each $\gamma(h)$ value by the square of the mean of those samples used to calculate that value. Alternatively, variograms can be computed on the logarithms of grade.

Commonly, values of $\gamma(h)$ increase steadily with increasing distance and reach a limiting or plateau level, as indicated in Fig. 12. The variograms in Fig. 12 demonstrate that nearby samples have similar grades and hence low variances, while others farther apart are quite different. This is an expression of the continuity in mineralization. An experimental variogram permits the interpretation of several characteristics of the mineralization as follows:

Range: The distance at which the variogram levels off at its plateau value is called the variogram range. This reflects the classical geologic concept of an area of influence. Beyond this distance of separation, sample pairs no longer correlate with one another and become independent.

Sill: $(C + C_0)$ The value where the variogram function plateaus is called the sill. For all practical purposes, the sill is equal to the variance of all samples used to compute the variogram. The difference between this value and the value of the variogram at distance (h) can be thought of as a level of certainty which exists when a sample assay is extrapolated ("extended," in geostatistics terms) over this distance. In other words, as the range is approached, the estimation variance based on extending sample grades increases to a maximum level of this sill.

Nugget Effect: (C_0) This is the name given to the variogram value $\gamma(h)$ at distance $h = 0.0$ m. It expresses the local homogeneity (or lack thereof) of the deposit. High nugget effect values relative to the sill can indicate that either the mineralization is poorly disseminated (i.e., tends to be concentrated in pockets or lenses), that the zone on which the variogram was computed was severely disjointed (e.g., major postmineralization structural discontinuities that exist in the deposit have been ignored), or that poor sample preparation and

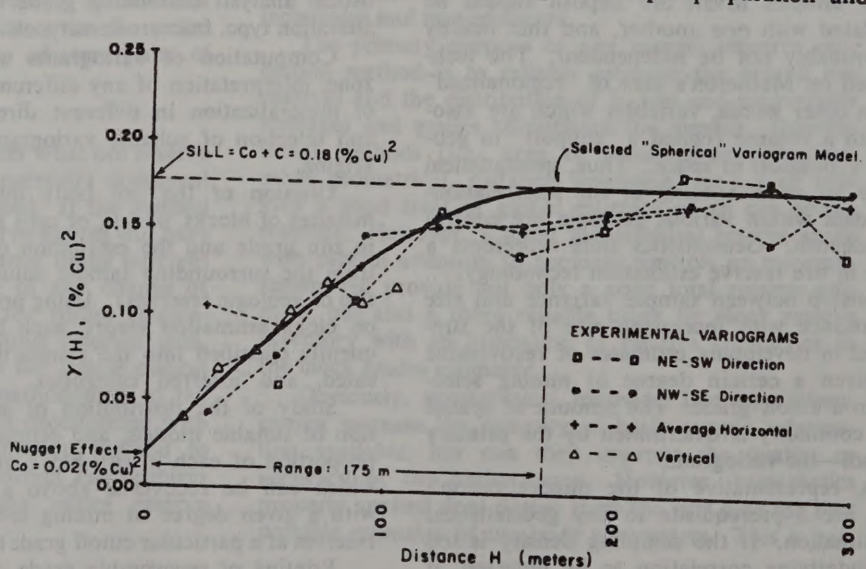


Fig. 12. Variograms for the secondary enrichment zone of a porphyry copper deposit based on a 15-m bench composite support.

assaying procedures were used. The latter item is called a "human" nugget effect.

Directional Anisotropy: This simply denotes whether the mineralization has greater continuity in a particular direction compared to another. The characteristic is analyzed by comparing the respective ranges of experimental variograms computed along different directions. Fig. 12 illustrates that for the deposit studied the mineralization was essentially isotropic, since the experimental variograms in alternate directions were very similar.

In practice, since drilling grids are rarely uniform, variograms are computed with a latitude on distance (i.e., half average squared differences between grades of sample pairs at a distance $h \pm \Delta h$ apart) and a spread or "window" on direction (i.e., all pairs falling within a direction of $\phi + \Delta\phi°$). Sample pairs are thus grouped into classes of Δh. This class size, the "window" angle on direction, and the sample or support size are very important parameters to choose correctly. They are often varied until a variogram showing the best structure is obtained.

To achieve reliable results, variograms should be computed on geologically similar zones of mineralization, or zones for which the mineralization had the same apparent genesis and for which there is no internal postmineralization faulting. If structurally continuous zones can be isolated, variograms can be computed independently for each, and then the individual results accumulated to produce an average variogram representative of the total mineralization. If major discontinuities are ignored, irregular distance distortions could occur which might lead to erratic or "noisy" variograms.

Poor variograms will also occur if there is only a limited amount of sample data, if the data are from a wide-spaced drilling program, or, finally, if the support chosen was too small. Obtaining good representative variograms involves a considerable amount of computation and interpretation by experienced geologists and geostatisticians. However, these efforts are amply rewarded through an enriched understanding of the deposit under study. An appreciation of the continuity in mineralization early in an exploration program is made possible by drilling a series of closely spaced "crosses" of holes, which can then be used to compute variograms. Such studies can be subsequently used to assist in planning an appropriate drilling grid which will permit ore reserve estimation to a certain required level of precision.

Ore Reserve Estimation Using Kriging

Kriging, the geostatistical grade interpolation method, calculates the grade of a block or panel as a linear combination of the grades of nearest samples. The coefficients of such a linear combination are obtained indirectly from the variogram. The method will yield the best estimates possible for in situ block grades, particularly where the sampling grid is very irregular and where the continuity in the mineralization is very different in alternate directions. Unlike other methods, kriging also gives a confidence level on each block estimate and on the overall reserves.

Given a series of spatially distributed drill holes or other samples, any ore reserve estimation method must obtain estimates for the average grade of blocks or panels between these drill holes. This is accomplished by some form of interpolation or extension technique, typically on a level-by-level or section-by-section basis. Because we are dealing with sample assays and not block assays, some error will be made in this estimation process. The "best" method will be one which keeps the errors as low as possible. Expressed in another way, if Z and Z^* are the true and estimated block grades, the variance of differences $(Z - Z^*)$ for all blocks must be minimized.

Kriging is simply a linear estimation method which develops optimal weights to be applied to each sample in the vicinity of the block being estimated. It uses both the position of the samples with respect to the block and the continuity of mineralization in different directions as portrayed by selected variogram models. The kriging estimator has the following general form:

$$Z^* = \sum_{i=1}^{n} \lambda_i x_i = \lambda_1 x_1 + \lambda_2 x_2 + \lambda_3 x_3 \ldots\ldots + \lambda_n x_n$$

where Z^* is the estimated in situ block grade, x_i is the sample grade in the vicinity of the block, λ_i is the weighting coefficient which will be assigned to respective x_i, and n is the selected number of nearest neighbor samples which will be used to estimate the block grade.

Even though we do not know the true block grade, Z, the estimation variance [i.e., variance of all $(Z - Z^*)$ errors] for such a linear combination of sample grades can be expressed as a function only of the variogram and of the weights λ_i. Therefore, having obtained a variogram(s) for a deposit, it is then possible to evaluate the estimation variance, and hence the confidence limits, which could be expected for any drilling grid and for any linear estimation method of determining block grades. This feature of analyzing the precision to be expected in estimating blocks of a given size and orientation using samples in a particular pattern around the block is unique to geostatistics. Classical statistical methods estimate confidence limits for a particular drill-hole spacing based on the standard error of the mean. This approach is only valid if samples are independent of one another, e.g., in an initial wide-spaced exploration grid.

Returning to the previous equation of the linear estimator, it should be noted that an interpolation method involving inverse distance weighting is also of this form where the λ_i coefficients become:

$$\lambda_i = \frac{1/d_i^e}{\sum_{i=1}^{n} 1/d_i^e}$$

where d is the distance of the particular sample to the center of the block, and e is the selected distance weighting exponent.

Kriging is not directly linked with distance as are distance weighting methods. Sample-to-block ristances are calculated, but only as a means to determine the corresponding sample-to-sample and sample-to-block covariances from the variogram.

Kriging utilizes the three-dimensional locations of samples, i.e., their distances and directions from the block. Irregular drilling patterns and/or highly anisotropic ore bodies are much more reliably studied, using

geostatistics compared with other methods which only involve distance weighting.

Kriging involves the optimal selection of the weights λ_i in such a manner that the estimation variance is minimized and such that $\Sigma\lambda_i = 1$. In minimizing this estimation error or variance, kriging results in a series of simultaneous equations. The equations can be solved for each weighting factor, λ_i, given the position of the sample, the size of the block to be estimated, and a model of the variogram representative of the mineralization being studied. The kriging process is quite mechanical, though time-consuming even on a digital computer. The calculated λ_i are then multiplied by respective sample grades to give the in situ block grade estimate. The estimation error for each block is of course also given. These estimation errors will be higher in regions of low drilling density and lower, as one would expect, where the deposit has been extensively drilled with closer spaced holes.

Linear kriging is the most common of the geostatistical estimating techniques. Nonlinear estimators such as lognormal and disjunctive kriging have become available within the past few years for more advanced studies, particularly relating to the determination of recoverable ore reserves. Disjunctive kriging is substantially more complicated and more demanding of computer resources than linear kriging. In most applications, the use of linear or lognormal kriging to estimate large in situ reserve blocks, followed by an assumption that selective mining units are distributed normally or lognormally within the large blocks, will lead to very good estimates of recoverable reserves. Needless to say, this latter step is vital, since we are trying to estimate the ore tonnage within economic excavation limits and its

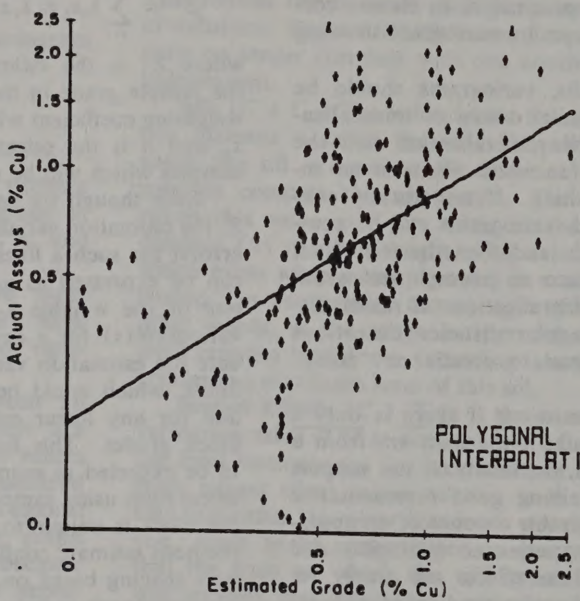
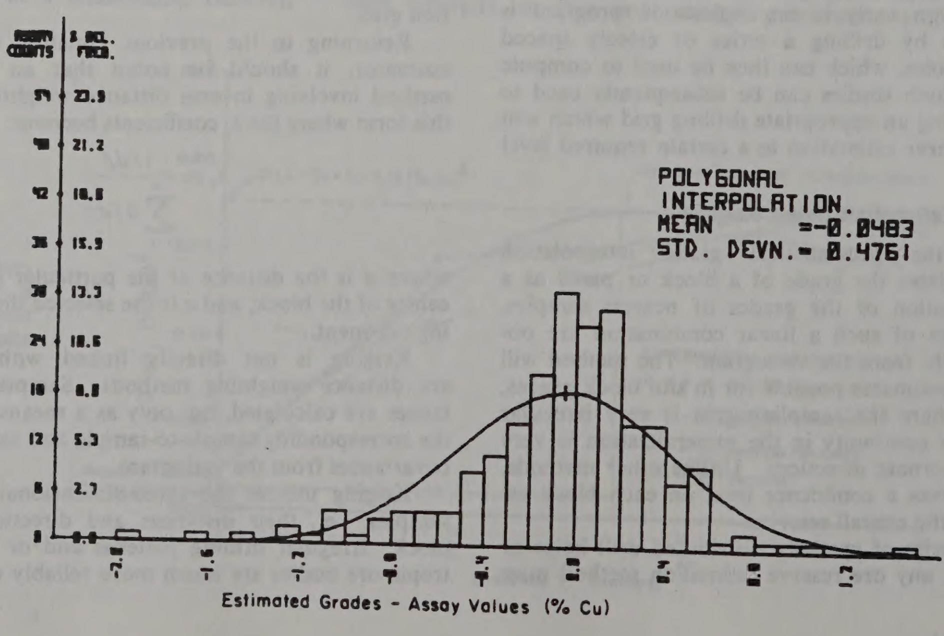


Fig. 13. Scatter diagram and error histogram for the polygonal interpolation method.



grade which will be mined and which will report to the mill.

Comparison With Other Methods

Understanding geostatistical ore reserve methods is a far more arduous task than the simpler technique involving area of influence type grade assignment used by the polygonal or other geometric methods. Geostatistics, like any of the more complex smoothing methods which utilize many samples in the vicinity of the block rather than only one, can only be economically achieved with a digital computer. Auditing of ore reserve calculations then becomes more difficult, as does one's ability to sensibly compare estimated block grades with the original sample values, because the former have a much reduced variability.

Given these problems and others, no matter what is said about the benefits of geostatistical techniques, there will always be doubts as to whether the added sophistication and cost is worthwhile, particularly in the absence of actual mining experience. During mining, of course, estimated block grades can be compared with reality. Such studies have been done at many operations in recent years, and the results offer substantial encouragement to the proponents of geostatistics. With such comparisons, however, additional doubts arise as to the representability of blasthole assays themselves and of their validity as an estimator of "true" block grade. Moreover, not all operations can afford the luxury of trial mining to confirm their grade prediction as part of a predevelopment feasibility study. As an alternative, the study described here provides a method of comparing various interpolation methods in the absence of actual mined grades in a quantitative way, rather than in terms of subjective comparisons.

This study was undertaken for the same porphyry

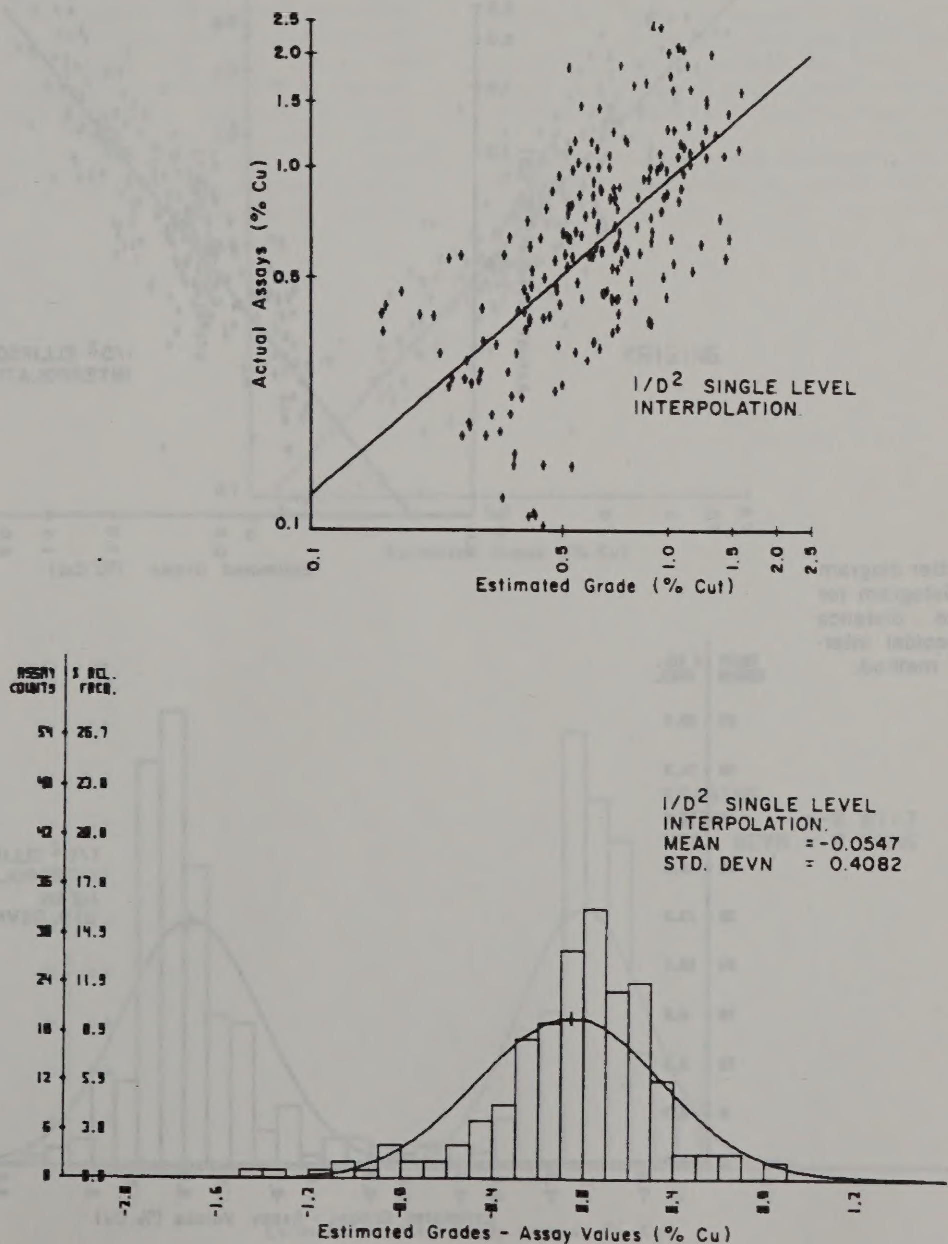


Fig. 14. Scatter diagram and error histogram for the inverse distance square (single level) interpolation method.

copper deposit for which the variograms of Fig. 12 were calculated. It involved the comparison of point estimated grades with actual point samples. Drill-hole sample assays were first composited as bench averages weighted by sample length. The composite values were considered as so-called "actual values." Some 200 drill-hole composites were successively removed from the total data set, and the same point in space was estimated from surrounding point assay values using a variety of methods: nearest sample grade assignment (polygonal approximation), inverse distance square interpolation using both horizontal and ellipsoidal searches, and finally linear kriging.

The removed holes were then "redrilled" as it were, and the estimated composite values were compared with true assays through the use of linear regression. The estimation method yielding a regression line which had a slope closest to 0.78 rad (45°), and which passed closest to the origin with minimum variability or scatter about the best fit line would be favored as the preferred methodology.

For each estimation method, the differences between the true assay values and estimated grades were also computed and plotted in the form of a histogram of errors. Once again, the best method would be the one which has a mean error of zero and the least variance on this error curve. Figs. 13 through 16 give the results of these studies for each of the methods examined.

Even for this large, relatively uniform isotropic porphyry copper deposit, it will be seen from Fig. 13 that the polygonal method more often than not did not estimate the true composite grade correctly. In general, the ellipsoidal search inverse distance square approach (Fig. 15) gave much better results than the horizontal case (Fig. 14). This was expected from the variogram study, which indicated that the mineraliza-

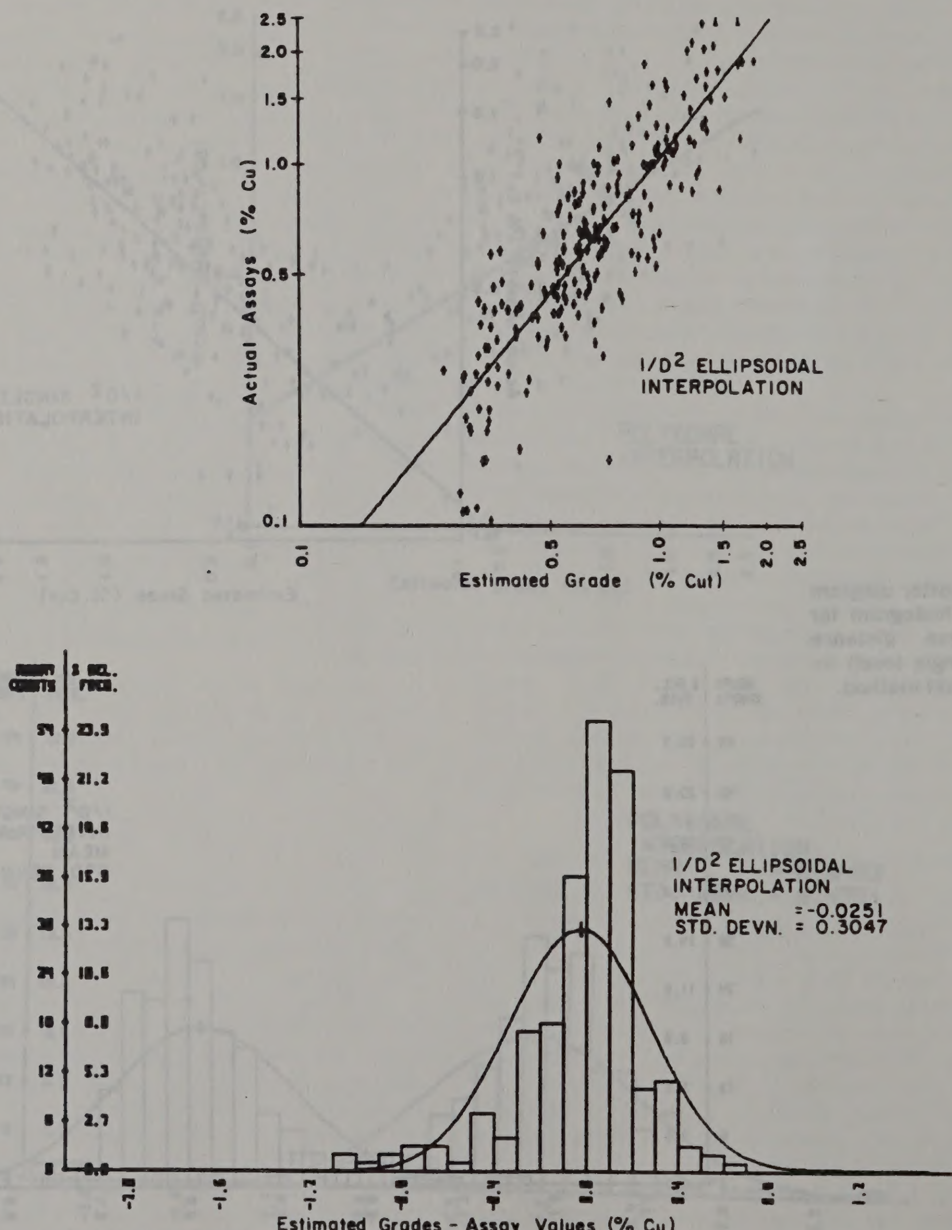


Fig. 15. Scatter diagram and error histogram for the inverse distance square ellipsoidal interpolation method.

tion was almost as continuous vertically as it was horizontally. Thus, samples directly above and below a block should have just as much "right" to be used in predicting the block grade as those at the same distance on the same level. Figs. 15 and 16 illustrate that there was a further marginal improvement in adopting linear kriging over the ellipsoidal inverse distance square method. It will be noted that prior knowledge of the variograms to define appropriate ellipsoidal search dimensions significantly enhances the performance of such methods, as is indicated here. Modern computer algorithms permit the user to vary both the search volume dimensions and the distance weighting exponent independently in each of three orthogonal directions to simulate anisotropic continuity in the mineralization. Under these circumstances, very good results can often be obtained, sometimes at less cost, using this less expensive, ellipsoidal inverse distance weighting approach rather than kriging.

Summary Remarks

Geostatistics provides a comprehensive suite of tools which must be tempered by an experienced ore reserve geologist/engineer to obtain reliable estimates—not only for the total ore reserve parameters, but also for the distribution of recoverable grade throughout the deposit for mine planning and scheduling purposes. Specifically, the geostatistical ore reserve methodology, if applied wisely, can yield quantitative information regarding the underlying characteristics of the mineralization, which are often heavily masked by detail. It is these ore controls and characteristics of mineralization which historically the economic geologist has had to subjectively

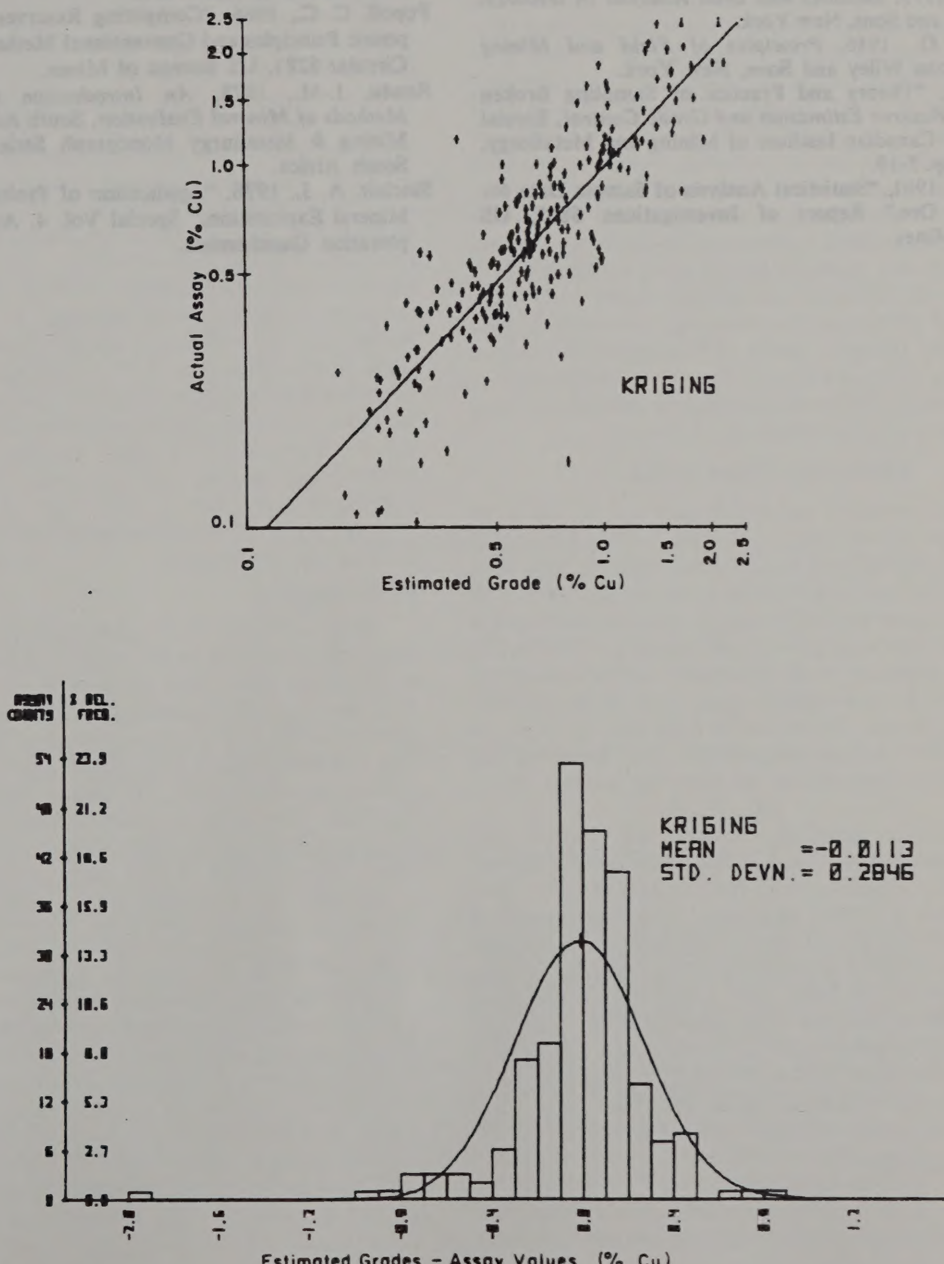


Fig. 16. Scatter diagram and error histogram for the kriging method.

define based on his experience with similar deposits. Having undertaken this interactive step of geological and variogram modeling of an ore body, one is then in a position to compute block grade estimates through kriging, to obtain an estimate of the probable error associated with each block grade, and to compute recoverable reserves based on smaller selective mining units above a certain cutoff grade.

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SAMPLING, DILUTION, AND RECOVERY

C. Alan Tapp

INTRODUCTION

Mine sampling determines the practicality of any mining operation. Improper sampling can result in an incorrect appraisal of present production and future potential. Therefore, the mine department in charge of ore-reserve calculations and mine sampling should be overseen by competent and experienced professionals with technical backgrounds qualifying them to produce accurate results.

Sampling is a process by which portions of an ore body are collected and analyzed to estimate the average mineral content of the entire ore body. It is incorrect to assume that a large number of samples eliminates any errors in the sampling method. To obtain unbiased samples, proper sample location with respect to rock type and mineralization is just as important. The sampling procedure must yield correct results for the type of mineral deposit, and careful consideration should be given to whether or not the sampling technique has been developed to an extent sufficient to eliminate as much human error and bias as possible. Only after the ore has been mined and milled is the sampling accuracy known.

Sampling also provides information about the bulk composition of the ore for mineralogical and metallurgical tests that determine the economic ore-waste boundaries and the geologic trends for exploration. Actual mining plans can be developed from this information to maximize profits. Accurate sampling is critical, and thus must be approached in a scientific manner.

SAMPLING TECHNIQUES

Sampling practices and techniques are as varied as the mines in which they are used. The method(s) chosen must be tailored to suit the company and mining needs. For instance, tabular uranium deposits, vein gold deposits, and porphyry copper deposits pose special problems in conducting unbiased sampling. The mine geologist or engineer in charge must develop a sampling method, test it in a sample area, and then critically evaluate the results. If the results from the test area are accurate within the economic limits established by the company, they then may be adopted for general use in the mine.

Four routine sampling methods are suitable for specific sampling objectives in the daily mine routine: (1) channel sampling, (2) chip sampling, (3) grab sampling, and (4) bulk sampling. The final sampling results depend upon how the four methods are combined to accurately determine the grade of the ore body. When used in conjunction with each other during different stages of mine development, the channel, chip, grab, and bulk sampling methods provide an in-house check or a comparison by which mining methods and sampling procedures can be evaluated. However, the most valid check is based upon the daily mill production.

Channel Sampling

Channel samples consist of cuttings collected from a groove cut into the rock about 102 mm (4 in.) wide and 19 mm (0.75 in.) deep. Various tools ranging from a 1.8-kg (4-lb) hammer and moil to a pneumatic chisel can be used to cut the sample. Accessibility and rock hardness determine the applicable sampling tools.

Before attempting to take a sample, the rock surface must be cleaned thoroughly; the method of cleaning depends upon the amount of mine dust accumulated on the surface or the degree of alteration of the rock surface. Typical cleaning methods employ a wire brush, water, or chipping a fresh surface.

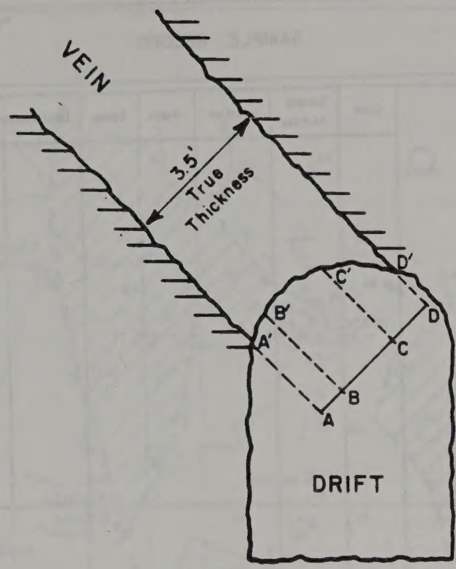
Next, the sample outline is marked on the prepared surface, taking care to choose appropriate sample locations. After determining the proper sample outline and length, the sample can be chiseled out, catching the rock fragments on a canvas tarpaulin on the floor, in a powder box, in a canvas bag to avoid contamination, or by some other suitable means to capture all of the sample. Vertical veins present a special problem because the drift's back usually is arched, not square. For example, Fig. 1 shows that if sample intervals are measured from A′ to D′, the length of the sample interval is greater than the true vein width, resulting in an incorrect calculation of the ore reserves. The correct method would collect three samples at A′-B′, B′-C′, and C′-D′, using either actual measurements in the mine represented by A-B, B-C, and C-D or trigonometric calculations to determine the true ore thickness. Then, each sample can be weighted by grade and true thickness for the vein. Channel lengths usually are a maximum of 1.5 m (5 ft), and it is good practice to divide longer samples into smaller intervals according to structures, changes in rock types, or differences in rock hardnesses. The influence of those features on the mineralization then can be determined.

Chip Sampling

Chip sampling is a variation of channel sampling used when the rock is too hard to channel sample economically or when little variation in the mineral content indicates that this sampling method will yield results similar to those of channel sampling. Rather than cutting a channel in the rock, small chips are flaked off at regular intervals over the entire face or area being sampled. Care must be taken to assure that the sample is representative of variations in the rock hardness and type. This method is fast and useful in preliminary evaluations, but it should not be used for quantitative ore-reserve calculations.

Grab Sampling

Grab sampling is a fast method for double checking either channel or chip-sampling procedures, and, in some instances, mine production can be estimated from carefully taken grab samples. Grab sampling takes equal amounts of material at selected intervals over a mine dump, a muck pile, or from an ore car to estimate its mineral content. Generally, this method is not considered reliable. Many independent variables can affect this type of sampling process. Thus, if the ore occurs in the softer fraction and a proportional amount of the result-



CORRECT METHOD

Sample Interval	Thickness, ft.	Assay, oz. Au/ton	Product, Th x Assay
A - B	0.8'	0.09	0.07
B - C	1.5'	0.50	0.75
C - D	1.2'	0.13	0.16
	3.5'		0.98

AVERAGE GRADE = $\dfrac{\Sigma(n.\bar{x})}{\Sigma n}$

AVERAGE GRADE = $\dfrac{0.98}{3.5'}$

AVERAGE GRADE = 0.28 oz./ton over a true thickness of 3.5' feet.

INCORRECT METHOD

Sample Interval	Thickness, ft.	Assay, oz. Au/ton	Product, Th x Assay
A' - B'	1.0'	0.09	0.09
B' - C'	1.5'	0.50	0.75
C' - D'	1.7'	0.13	0.22
	4.2'		1.06

AVERAGE GRADE = $\dfrac{\Sigma(n.\bar{x})}{\Sigma n}$

AVERAGE GRADE = $\dfrac{1.06}{4.2'}$

AVERAGE GRADE = 0.25 oz./ton over an incorrect thickness of 4.2' feet.

Fig. 1. Calculation of a true sample thickness for mine sampling.

ing fines are not sampled, the results are erroneous. The sample may also consist more of one rock color, rather than having the correct proportions of each. If each sampler is consciously aware of every variable that can affect grab sampling, the reliability of the method increases.

Bulk Sampling

Bulk sampling is used to evaluate ore zones with irregularly distributed mineralization or to determine composite ore characteristics. The amount collected depends upon the nature of the mineralization and the size of the area being tested. A bulk sample can range from tens of kilograms (pounds) to several tons. For example, if a mine dump is to be evaluated economically, it is necessary to take several bulk samples at predetermined locations, or several small samples can be combined into one bulk sample. Then, weighted averages of the assays, based upon the portion of the mine dump they represent, are combined to obtain a representative value for the dump. As another example, if the accuracy of channel samples from a vertical stope is questionable, the area can be bulk sampled before mining. In this case, several holes drilled into the back are shot, with the sample collected on a canvas tarpaulin. A sample like this could amount to several hundred kilograms (pounds). However, care must be taken to avoid "salt-

ing" or otherwise influencing the results through poor sampling techniques.

MINE SAMPLING

In the day-to-day operations of a mine, the importance of accurate sampling sometimes can become obscured by the routine nature of the job. However, this daily information is necessary for the mining department to develop accurate mine-production plans. Such daily sampling provides information for four phases of mining: guiding the development headings, providing data for ore-reserve calculations, indicating production grades for every stope, and allowing mineralogical and metallurgical evaluation of the ore body.

Development Headings

All development headings are sampled on a daily basis to determine the ore content in the mining face. This determines the assay boundaries and helps in the projection of mill head grades. Customarily, channel or chip samples are taken at the face, provided the face has been cleaned from the previous round; the ore characteristics will determine the best type of sampling method. If the face is not clean, grab samples may be taken from the muck pile; although grab samples may not be as accurate as channel samples, they can be used with caution.

When the development headings are sampled, all pertinent information is recorded on work sheets immediately. The location of the sample should be referenced against some permanent point. The thickness of the sample intervals, rock type, sample type, sampling party, date, and sample number all should be recorded. As a general rule, daily work sheets are kept in loose-leaf notebooks so completed sheets can be left in the office. If the sample book is lost, only one day's records are lost, rather than the records for an entire month or year. An example of such a daily sample sheet is shown in Fig. 2.

Once in the office, the work-sheet data is then transferred to the permanent mine records. Fig. 3 shows a sample sheet for these records. The permanent sample records are no more than a summation of the daily sampling, prepared in concise and easily recognizable form. In most cases, that means the creation of a tabular form for the files. Accurate records are essential for maintaining the mine's head grade; they are the key to mine

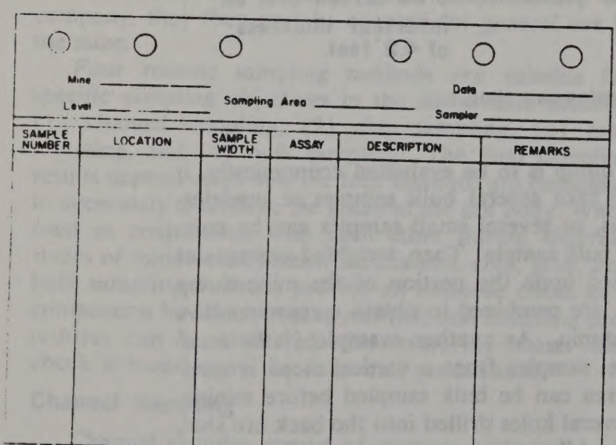


Fig. 2. Example of a daily sample sheet.

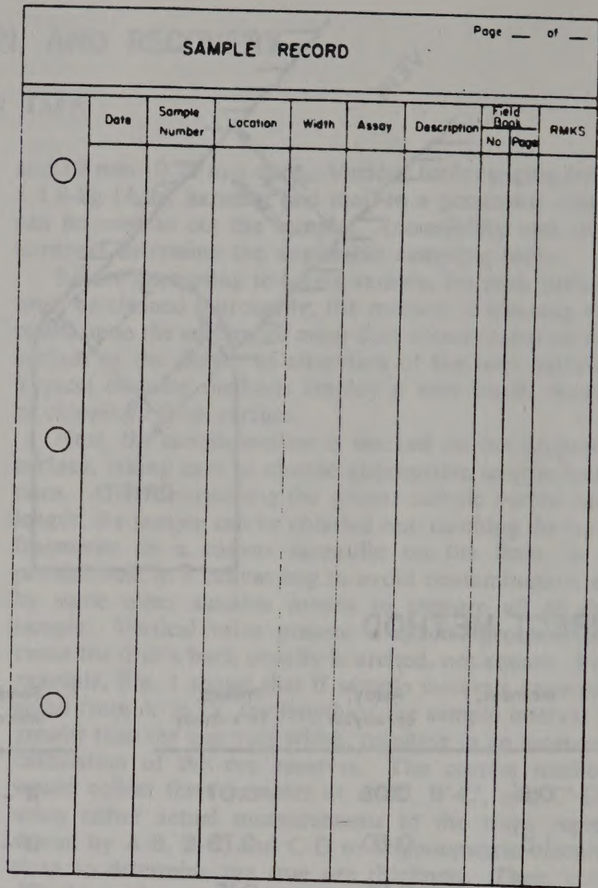


Fig. 3. Form to record daily sampling for permanent mine records.

expansion and development of potential exploration targets.

Assay maps and cross sections of the appropriate scale for the area also are updated with the new sample information. They are the most usable form for evaluating daily sampling information. The maps and cross sections present a visual picture of the limits and orientation of the ore. As shown in Figs. 4 and 5, it is customary to have two sets of mine maps showing both small-scale and large-scale features. Small-scale mine maps at 25 mm = 61 m (1 in. = 200 ft) are useful in showing mine-development plans, gross geological features, property boundaries, and general ore trends; a generalized overview of the mine property is shown on these small-scale maps. The large-scale maps at 25 mm = 15 m (1 in. = 50 ft) are used to plot the daily sampling results and mining plans. On these large-scale maps, daily mine face advancement is determined more accurately. They also provide excellent working maps for the mine geologist. Features not large enough to be plotted on a small-scale map, but important to the interpretation of the deposit, are plotted on the large-scale map.

Cross sections also are made at two scales for similar reasons. Fig. 6 shows both a small-scale longitudinal section and a plan-view map of a hypothetical mine.

Ore-Reserve Calculations

Ore-reserve data are parts of the permanent mine record and should be updated on a periodic basis to include newly developed reserves and to reflect any

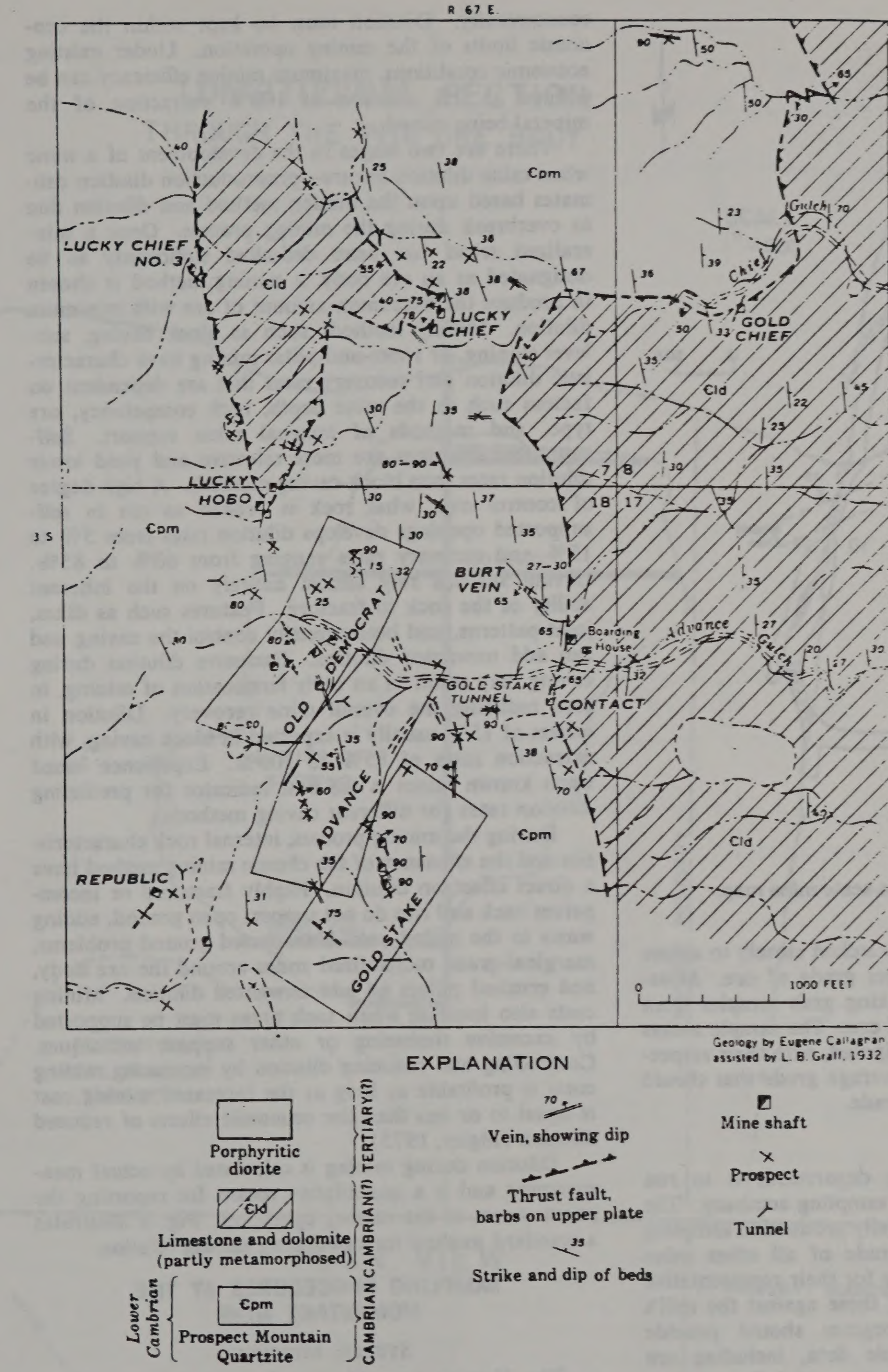


Fig. 4. Small-scale geologic map of the Chief district.

depletion of reserves from stoping areas. For most mines, once the ore has been blocked out, the perimeter of each pillar is sampled on equally spaced centers. Large-scale assay maps are produced from these sample data. When the mine maps are in the same plane as the ore body, pillar areas can be planimetered to estimate tonnages accurately. Weighted averages for all of the sample points are calculated to produce ore grades for the pillar, as shown in Fig. 7. Because ore minerals usually are not distributed homogeneously throughout an ore body, it may be necessary to spot check the sample locations, a task easily accomplished by bulk sampling the points where channel samples are in doubt.

Production Grades

Day-to-day sampling procedures provide sample control for daily production. Mine-head grade, as

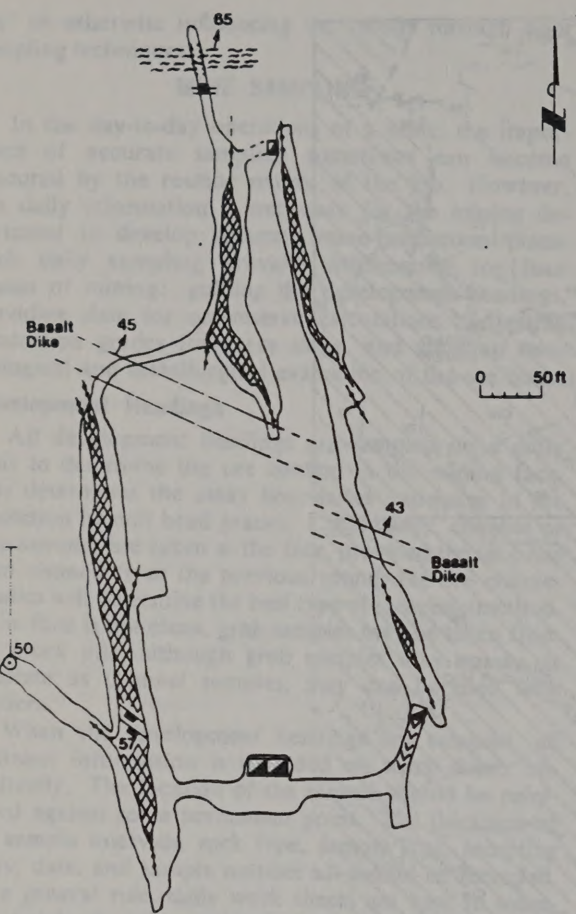


Fig. 5. Example of a large-scale mine map.

opposed to mill-head grade, is watched closely to assure that the mill receives the proper grade of ore. Mine-head grade is monitored by taking grab samples from every car, train, or shipment of ore. The sample assays for a working day then are computed, with their respective tonnages, for a weighted average grade that should be an approximate mine-head grade.

Internal Checks

One duty of the sampling department is to run internal checks on the in-house sampling accuracy. The simplest method is to compare daily production sampling with a calculated composite grade of all other mine samples (weighted appropriately for their representative proportions), finally comparing these against the mill's production. The sampling program should provide enough varied forms of sample data, including ore reserves, so that different combinations of daily sampling data can be used to determine the accuracy of the program. There never can be enough samples; the information that they provide is essential to meet the production quotas.

DILUTION AND RECOVERY

In a mining context, dilution is the extracted tonnage of material below the economic cutoff grade for the mine. The extracted material may contain the mineral being mined but in insufficient quantities to be recovered economically. Dilution must be kept within the economic limits of the mining operation. Under existing economic conditions, maximum mining efficiency can be defined as 0% dilution at 100% extraction of the mineral being mined.

There are two stages in the development of a mine when mine dilution occurs—preproduction dilution estimates based upon the mining method and dilution due to overbreak during the mining process. Once a mineralized trend has been delimited sufficiently to be designated as an ore body, a mining method is chosen to produce the maximum amount of ore with minimum dilution. Mining methods such as block caving, sublevel stoping, or room-and-pillar mining have characteristic dilution and recovery rates that are dependent on factors such as the mine depth, rock competency, ore type, and methods of internal mine support. Self-supported openings are more selective and yield lower dilution rates than block-caving methods. A high degree of control over what rock is broken as ore in self-supported openings develops dilution rates from 5% to 15% and recovery rates ranging from 60% to 85%. Caving systems rely almost entirely on the inherent ability of the rock to fracture. Features such as dikes, joint patterns, and heterogeneity control the caving and can add unwanted dilution. Excessive dilution during caving may result in an early termination of mining, in turn reducing the overall mine recovery. Dilution in excess of 15% usually is expected in block caving, with extraction rates of 85% to 100%. Experience based upon known mines is the best indicator for predicting dilution rates for different caving methods.

During the mining process, internal rock characteristics and the efficiency of the chosen mining method have a direct effect on dilution. Highly fractured or incompetent back and ribs do not support open ground, adding waste to the mined ore. Unexpected ground problems, marginal-grade mineralized zones around the ore body, and crushed pillars all add unwanted dilution. Mining costs also increase when such areas must be supported by excessive timbering or other support techniques. Controlling and reducing dilution by increasing mining costs is profitable as long as the increased mining cost is equal to or less than the economic effects of reduced dilution (Ingler, 1975).

Dilution during mining is calculated by actual measurement and is a quantitative means for reporting the effectiveness of the mining operation. Fig. 8 illustrates a standard method for calculating mining dilution.

SAMPLING PROCEDURES AT THE HOMESTAKE MINE

STEVEN MITCHELL

The Homestake gold mine is located in the Black Hills, in Lawrence County, at Lead, SD.

General Geology

During the Precambrian era, an accumulation of mudstone was deposited and later highly refolded and transformed by regional metamorphism. Extreme compressional effects resulted in a plastic-type flow of the rocks, producing a series of anticlines and synclines, locally known as "ledges." Fig. 9 illustrates a cross section of the ledge structure. After much erosion, sediments of the Paleozoic and Mesozoic eras accumu-

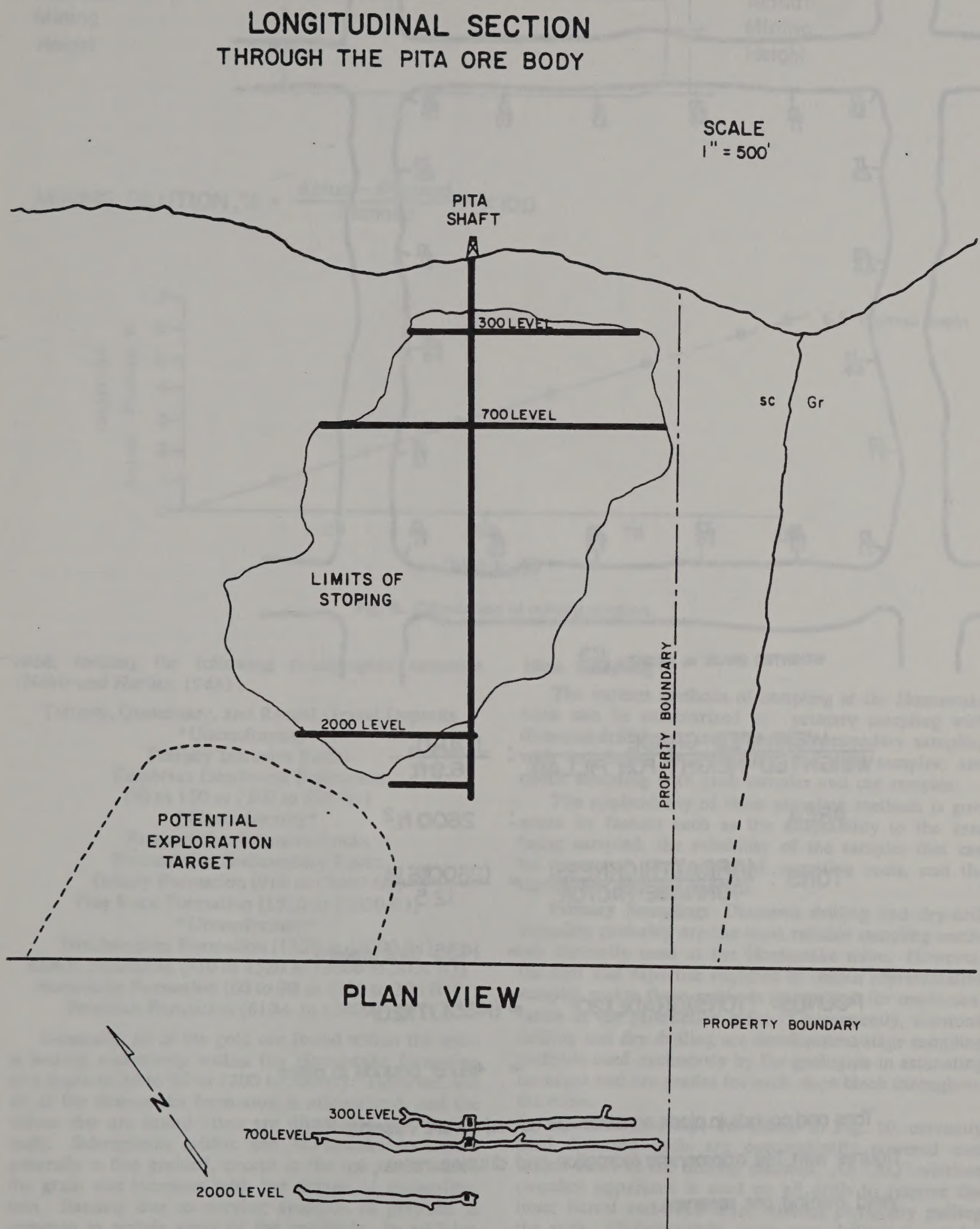
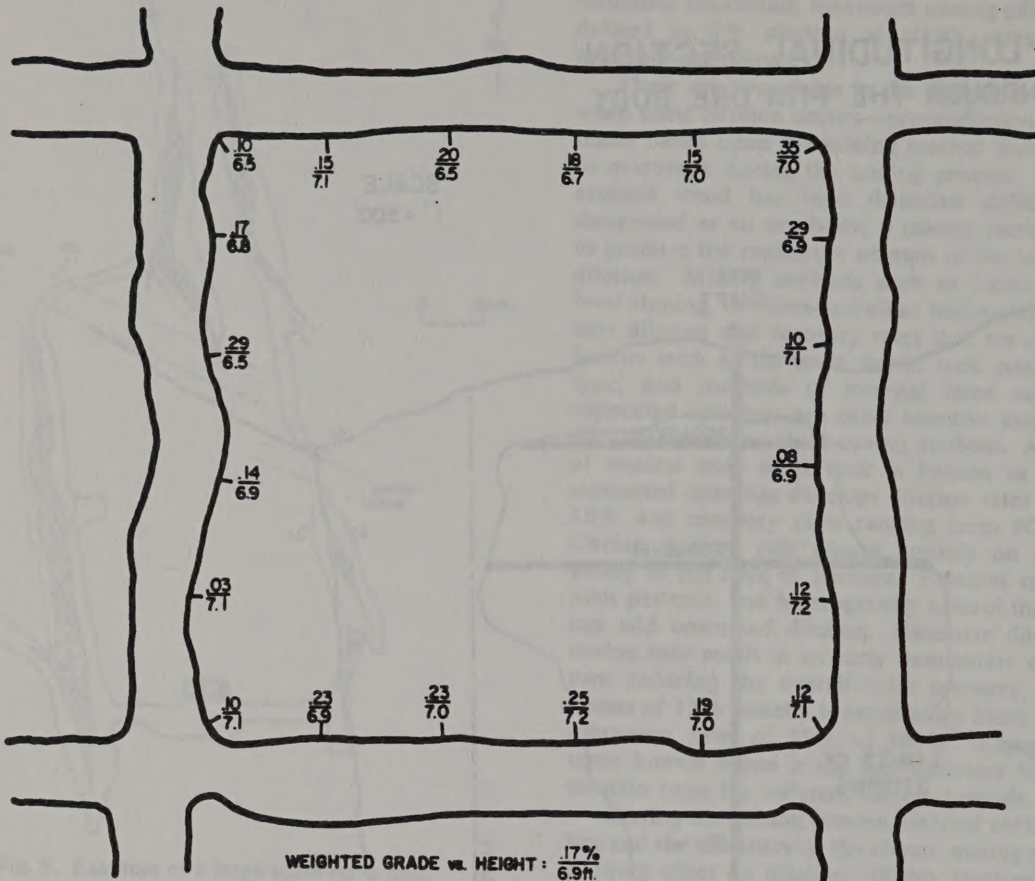


Fig. 6. A small-scale longitudinal section with accompanying plan view.



$$\frac{\text{WEIGHTED GRADE}}{\text{WEIGHTED HEIGHT FOR PILLAR}} : \frac{.17\%\,U_3O_8}{6.9\text{ ft.}}$$

AREA : 2600 ft.^{2}

$$\text{TONS} : \frac{(\text{AREA})(\text{THICKNESS})}{\text{TONNAGE FACTOR}} = \frac{(2600)(6.9)}{12.5}$$

= 1435 tons in place

POUNDS: (TONS)(GRADE)(20) = (1435)(.17)(20)

= 4879 pounds in place

Tons and pounds in place can then be added to ore reserve figures with the appropriate extraction and dilution rates, to yield mining ore reserves.

Fig. 7. Ore reserve calculations from a large-scale mine map.

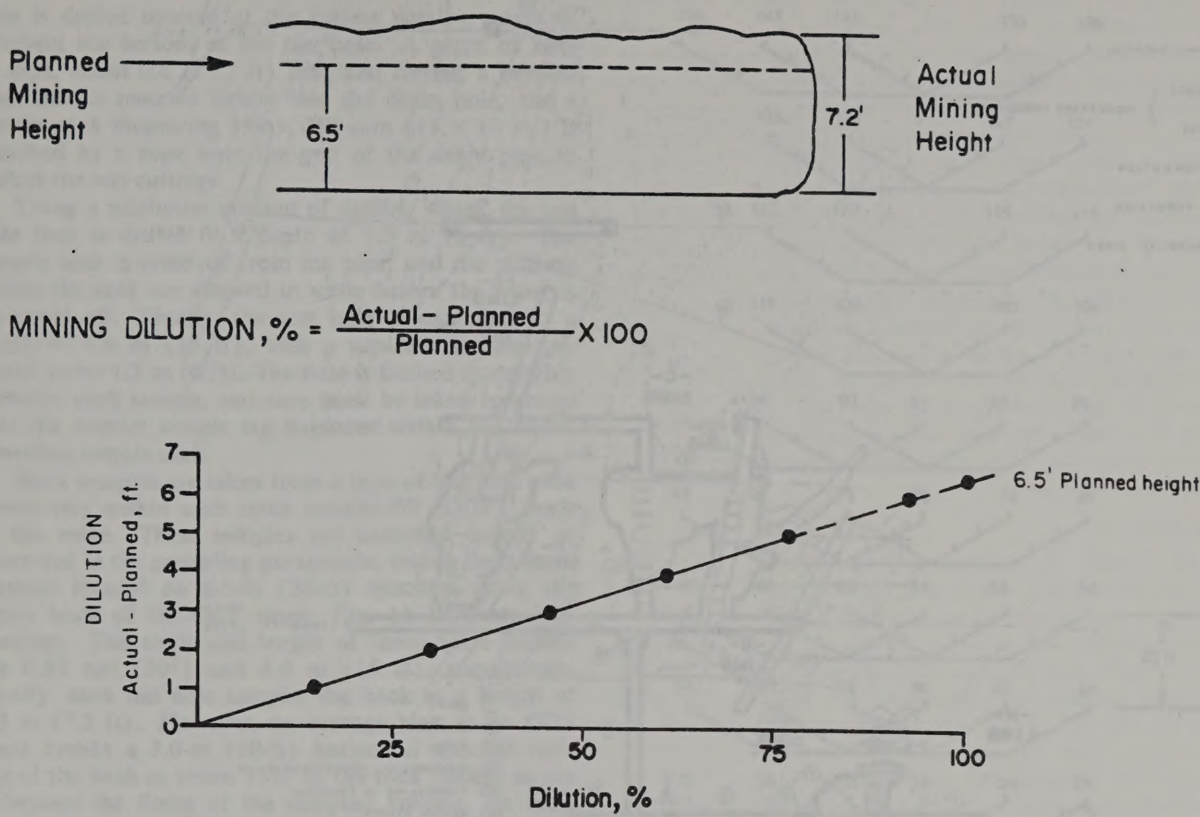


Fig. 8. Calculation of mining dilution.

lated, forming the following stratigraphic sequence (Noble and Harder, 1948):

Tertiary, Quaternary, and Recent Gravel Deposits
Unconformity
Tertiary Intrusive Rocks
Cambrian Deadwood Formation
[90 to 150 m (300 to 500 ft)]
Unconformity
Precambrian Intrusive Rocks
Precambrian Sedimentary Rocks
Grizzly Formation [910 m (3000 ft)]
Flag Rock Formation [1520 m (5000 ft)]
Unconformity
Northwestern Formation [1220 m (4000 ft)]
Ellison Formation [910 to 1520 m (3000 to 5000 ft)]
Homestake Formation [60 to 90 m (200 to 300 ft)]
Poorman Formation [610+ m (2000+ ft)]

Generally, all of the gold ore found within the mine is located exclusively within the Homestake formation at a depth of 60 to 90 m (200 to 300 ft). However, not all of the Homestake formation is mineralized, and the values that are found often are distributed very erratically. Sideroplesite within the Homestake formation generally is fine grained, except in the ore zones where the grain size increases with the degree of mineralization. Banding due to varying amounts of graphite is common in certain areas of the ore body. In addition to sideroplesite and graphite, the Homestake formation contains quartz, biotite, chlorite, pyrite, chalcopyrite, pyrrhotite, arsenopyrite, and ankerite. The gold content averages approximately 5 g/t (0.2 oz per st).

Mine Sampling

The various methods of sampling at the Homestake mine can be summarized as: primary sampling with diamond-drill holes and dry drills; secondary sampling with test holes, drill samples, and pick samples; and check sampling with grab samples and car samples.

The applicability of these sampling methods is governed by factors such as the adaptability to the area being sampled, the reliability of the samples that can be recovered, the associated operating costs, and the duration of any one method.

Primary Sampling: Diamond drilling and dry-drill sampling probably are the most reliable sampling methods currently used at the Homestake mine. However, the cost and expertise required to obtain representative samples makes these methods impractical for implementation at the production stage. Consequently, diamond drilling and dry drilling are development-stage sampling methods used exclusively by the geologists in estimating tonnages and ore grades for each stope block throughout the mine.

Diamond Drilling—As shown in Fig. 10, currently used diamond drills are pneumatically powered and utilize water as the drilling medium. An "AQ" wireline overshot apparatus is used on all drills to remove the inner barrel and rock core, without physically pulling the rods. Unfortunately, such core drilling is a relatively slow and expensive process. Although distances of up to 58 m (190 ft) have been reported, progress of 12 m (40 ft) per 8.5-hr shift per drill crew is considered to be average. The total operating costs for AQ wireline

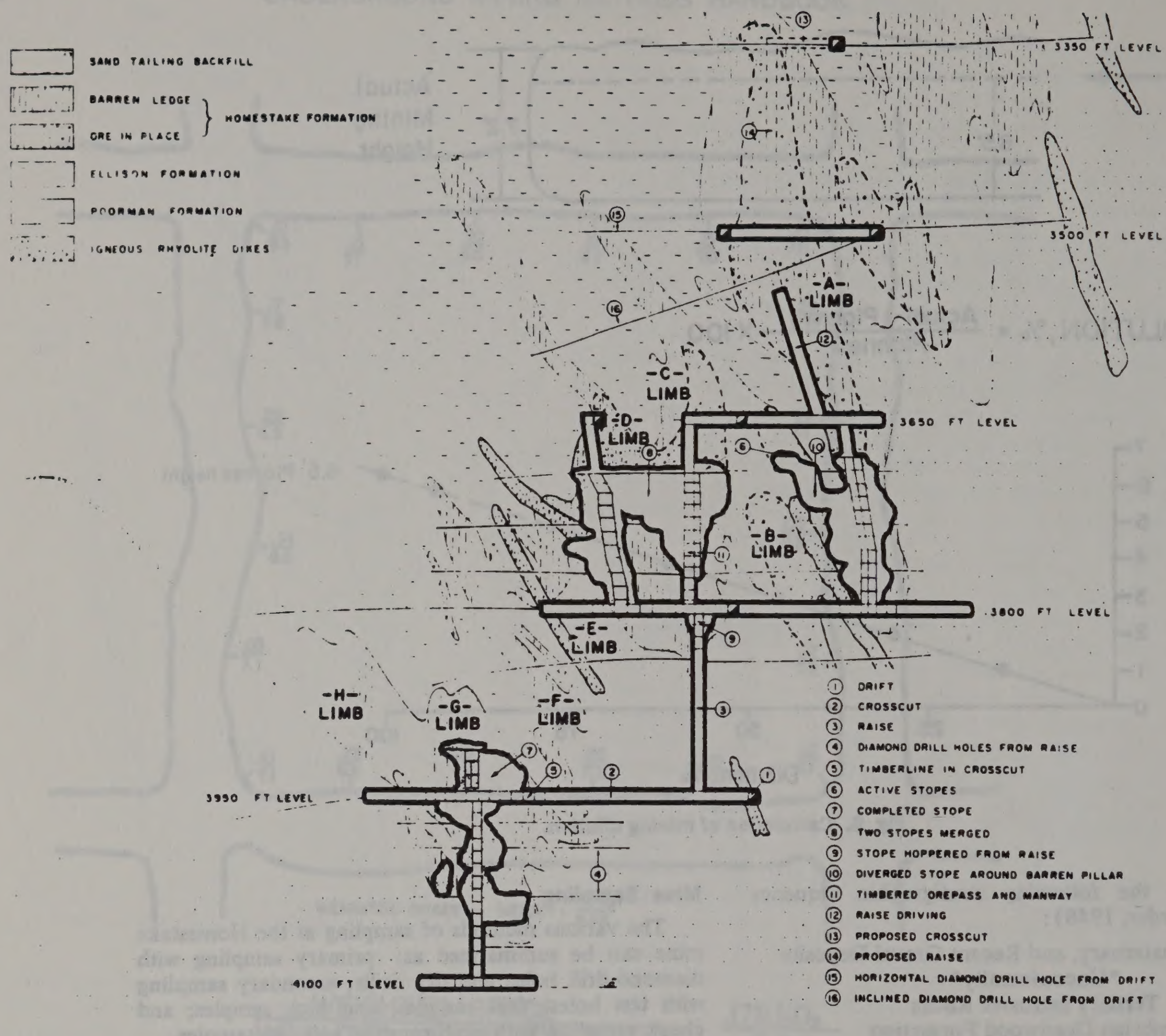


Fig. 9. Cross section of a nine-ledge structure.

drilling are approximated at \$18.04/m [\$5.50 per ft (1977 US \$)].

Dry Drilling—For several years, the LeRoi Dry Ductor drill (LLV) has been used as a primary sampling device. The recovery of dry-drill cuttings is about 90%. Because the cuttings are drawn back through the drill rods, a special 25-mm (1.0-in.) drill rod is required in conjunction with a 41-mm (1.625-in.) Vacumatic® bit. Dry-drilling sample data are used by the geologists to supplement the diamond-drill data in estimating the ore reserves.

Secondary Sampling: Test holes and drill samples both involve collecting the wet drill cuttings or sludge from percussion jackleg drills. These two sampling methods are used extensively within the active stoping blocks that have been outlined by previous diamond-drill and dry-drill sampling procedures.

Test Holes—To obtain a representative sample at the mining fact, test holes are drilled across the "grain" of the ore-bearing Homestake formation. To begin the process, the collar of the test hole is drilled approximately 254 mm (10 in.) into the wall. Then, a drain

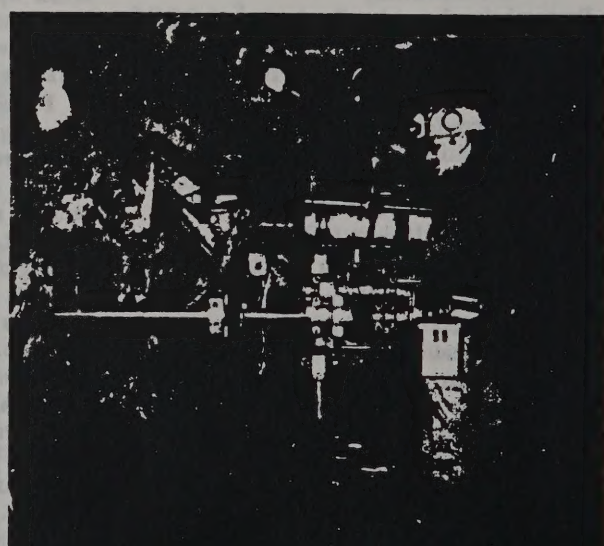


Fig. 10. An air-powered AQ wireline drill.

hole is drilled upward at the flattest possible angle to intersect the bottom of the test hole. A piece of hose or pipe, about 0.6 m (2 ft) long and having a beveled end, then is inserted tightly into the drain hole, and a canvas sack measuring 356 × 254 mm (14 × 10 in.) is attached by a rope onto the end of the drain pipe to collect the wet cuttings.

Using a minimum amount of drilling water, the test hole then is drilled to a depth of 1.5 m (5 ft). The sample sack is removed from the pipe, and the cuttings within the sack are allowed to settle before the water is decanted off. Finally, the test hole is extended to a depth of 4.6 m (15 ft), with a separate sample collected every 1.5 m (5 ft). The hole is flushed thoroughly between each sample, and care must be taken to assure that the correct sample tag is placed within the corresponding sample sack.

Back samples are taken from a type of test hole used extensively within each open cut-and-fill (OCF) stope in the mine. These samples are collected exactly as described in the preceding paragraphs, taking them from sections located on 6.1-m (20-ft) spacings along the entire back of the OCF stope; Fig. 11 illustrates the spacings. The angle and length of these holes usually are 0.52 rad (30°) and 4.6 m (15 ft), respectively. Ideally, each test hole samples the back to a height of 2.3 m (7.5 ft). However, an average blast in an OCF stope breaks a 3.0-m (10-ft) horizontal slice of rock out of the back so about 33% of the rock broken as ore is beyond the limits of the sampled volume. In most stopes, the sample density, as shown in Fig. 12, is approximately 66 t (73 st) per ore sample.

At the ends of each row or section of sample holes, additional 4.6-m (15-ft) test holes often are drilled into the footwall and hanging wall. These holes are drilled approximately 50% of the way between the floor and the end back-sample test hole in that row. In addition to the back samples and the wall samples, each stope usually contains at least 10 to 15 other test holes. Those holes generally are used to check back-sampled areas that are of questionable ore grade or to test for possible ore extensions within the areas of the stope walls.

Drill Samples—Drill samples are similar to those taken from test holes, except that the cuttings from an entire hole are collected, rather than from separate and distinct intervals within the hole. If a drift round is to be sampled by a drill hole, the cuttings normally are allowed to run onto the floor of the drift until the round is completed. The cuttings then are loosened with a pick and placed into a sample sack, along with an identifying tag. Often, an extension hole is drilled to sample the rock ahead of the round that presently is being drilled. In that case, the cuttings are collected in the same manner as for a test hole.

Pick Samples—If properly taken, the pick sample is reliable, and it is used in conjunction with test holes and drill samples. Pick samples are used at the faces of drift rounds, slab rounds, and raise rounds. A typical area to be sampled usually is about 0.3 m (1.0 ft) wide and up to 2.4 m (8.0 ft) long. Small pieces of rock are chipped from within this area, taking care to remove equal amounts of rock across the grain and across the entire face of the drift or raise round to assure a reliable and representative sample. A pick sample does not relate to the size or extent of the body of rock behind the face.

Check Sampling: Check sampling serves only to indicate rock grade that has been mined previously and is now lying in the broken state. A sufficient portion of the rock is gathered at random to constitute a representative sample. Such a sample is a valuable tool for

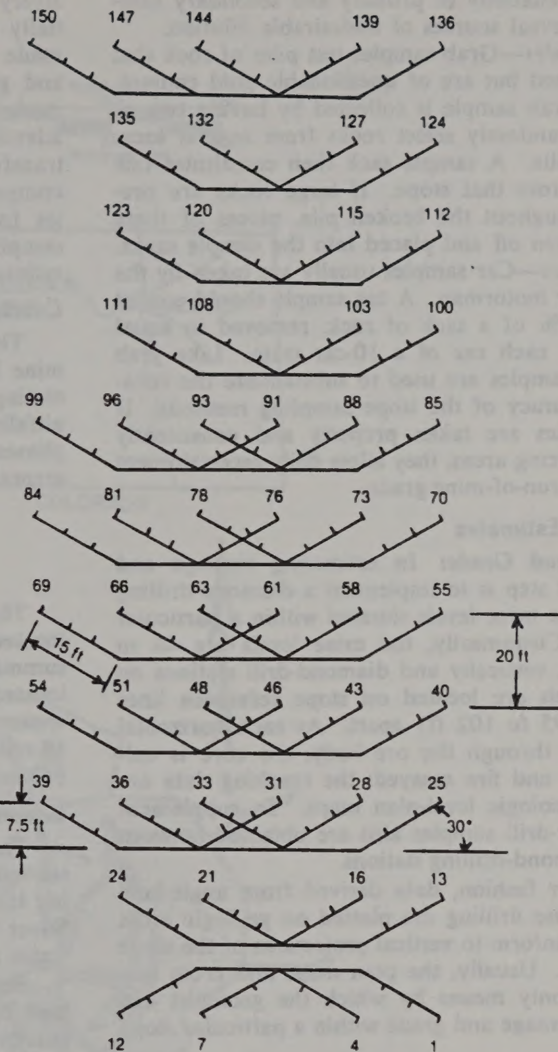


Fig. 11. Typical map showing test-hole pattern in an open cut-and-fill (OCF) stope (1 ft = 0.3048 m).

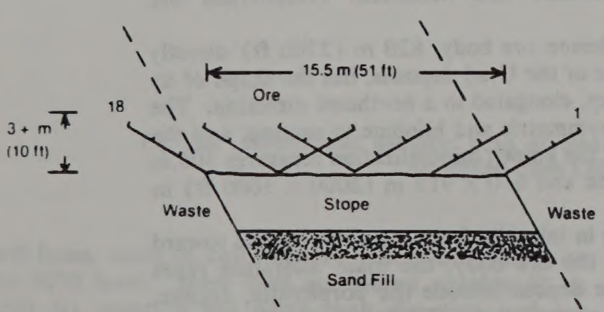


Fig. 12. Typical cross section of an open cut-and-fill (OCF) stope showing sample density (51 x 10 x 20 ft = 10,200 cu ft; 10 cu ft/st = 1020 st; 1020 st/14 samples = 73 st/sample).

checking the reliability of primary and secondary sampling and to reveal sources of undesirable dilution.

Grab Samples—Grab samples test piles of rock that have been mined but are of questionable gold content. Normally, a grab sample is collected by having two or three people randomly select rocks from several locations on the pile. A sample sack then constitutes one grab sample from that stope. If large rocks are predominant throughout the broken pile, pieces of these rocks are broken off and placed into the sample sacks.

Car Samples—Car samples usually are taken by the chute puller or motorman. A car sample should consist of at least 75% of a sack of rock, removed in equal amounts from each car of a 10-car train. Like grab samples, car samples are used to substantiate the reliability and accuracy of the stope sampling methods. If the car samples are taken properly and consistently from all producing areas, they allow daily ascertainment of the average run-of-mine grade.

Ore Reserve Estimates

Tonnage and Grade: In estimating tonnage and grade, the first step is to implement a diamond-drilling program on the mine levels situated within a particular stope block. Customarily, the mine levels are 46 m (150 ft) apart vertically and diamond-drill stations on individual levels are located on stope reference lines 29 to 31 m (95 to 102 ft) apart. As each horizontal hole is drilled through the ore body, the core is collected, logged, and fire assayed; the resulting data are recorded on geologic level-plan maps. To supplement these data, dry-drill samples also are obtained between successive diamond-drilling stations.

In a similar fashion, data derived from angle-hole drilling and raise drilling are plotted on geologic cross sections that conform to vertical projections of the stope reference lines. Usually, the plan maps and cross sections are the only means by which the geologist can estimate the tonnage and grade within a particular stope block.

Once the sampling data have been plotted on maps and cross sections, a stoping outline can be established in accordance with the overall mine cutoff grade (determined by prevailing economic conditions). At present (1978), there are five techniques used for estimating tonnages, all of which are variations of the polygon method of calculating ore reserves. For example, an ideal situation would be one in which the stope outline conforms to a cross-sectional triangle. With such a configuration, the tonnage can be estimated using a "wedge" volume calculation.

To estimate the grade for this same stope block, a variable dilution factor would be incorporated into the calculation to compensate for hanging wall dilution, as well as for dilution from the hydraulically backfilled sand floor used in OCF stoping. The averages of the assays from the level plans and cross sections then would be weighted in relation to the size of the planimetered area in that plan or cross section. Finally, as with any estimate, the tonnage and grade would be upgraded constantly as test-hole samples became available from actual stope production.

Production Forecasting: Ultimately, the ore estimates from each stope block on every level are added together to arrive at a total figure for the ore reserves. Every month thereafter the ore estimates for each potentially productive stope are reviewed and a forecast is made to predict the upcoming month's ore production and grade for the entire mine. Similar forecasts are made for mine production six months and one year in advance. Then, check samples from every ore and waste transfer system—from the mine to the mill—can be compared to the calculations, allowing correlation studies for the reliability of the primary and secondary sampling methods and for the accuracy of the ore-estimating technique.

Conclusions

The sampling procedures used at the Homestake mine have proven to be a satisfactory method of estimating ore reserves. However, the system has many pitfalls, requiring constant and rigid supervision of all phases to avoid faulty data that would cause serious errors in the ore estimates and grade control.

SAMPLING PRACTICES AT THE HENDERSON MINE

BRUCE R. STANLEY

The Henderson molybdenite (MoS_2) deposit is located 1100 m (3600 ft) under the 3750-m (12,311-ft) summit of Red Mountain in Colorado. Red Mountain is located in the Daley mining district at the extreme western edge of the Colorado mineral belt and is 13 km (8 miles) west of the town of Empire and about 69 km (43 miles) west of Denver.

General Geology

The geology at Red Mountain consists of a classic subvolcanic rhyolite porphyry sequence emplaced during the mid-Tertiary period in multiphase Precambrian Silver Plume granite of the Front Range. Fig. 13 illustrates a generalized geologic map of the area.

Emplacement of the porphyries probably was localized by the intersection of major Precambrian faults reactivated in the Tertiary period. Igneous pulses, some never quite reaching the surface, cooled and crystallized into a pipe-shaped complex that consists of at least two breccia dikes, several rhyolitic dike systems, and six plugs. The earliest plug and the lower related stock complex probably were responsible for the molybdenite mineralization. Stockwork quartz-molybdenite mineralization occurs both in the extensively fractured granite and porphyry breccia near the top of Red Mountain (Urad ore body) and in the more coarsely crystalline Urad-Primos porphyry Henderson-granite stock complex deep beneath Red Mountain (Henderson ore body).

The Henderson ore body, 820 m (2700 ft) directly below the base of the Urad deposits, has the shape of an inverted teacup, elongated in a northeast direction. The ore body is asymmetric and bilobate in section, and the total extent of the known mineralization measures 305 m (1000 ft) thick and 610 × 915 m (2000 × 3000 ft) in plan.

Increasing in intensity from the outer fringes toward the center of the ore body, the main alteration types enveloping the deposit include the porphylitic, argillic, quartz-sericite-pyrite, and potassic zones. Ten alteration zones have been identified at the Henderson mine, as shown in Fig. 14.

The ore body is sampled by underground diamond

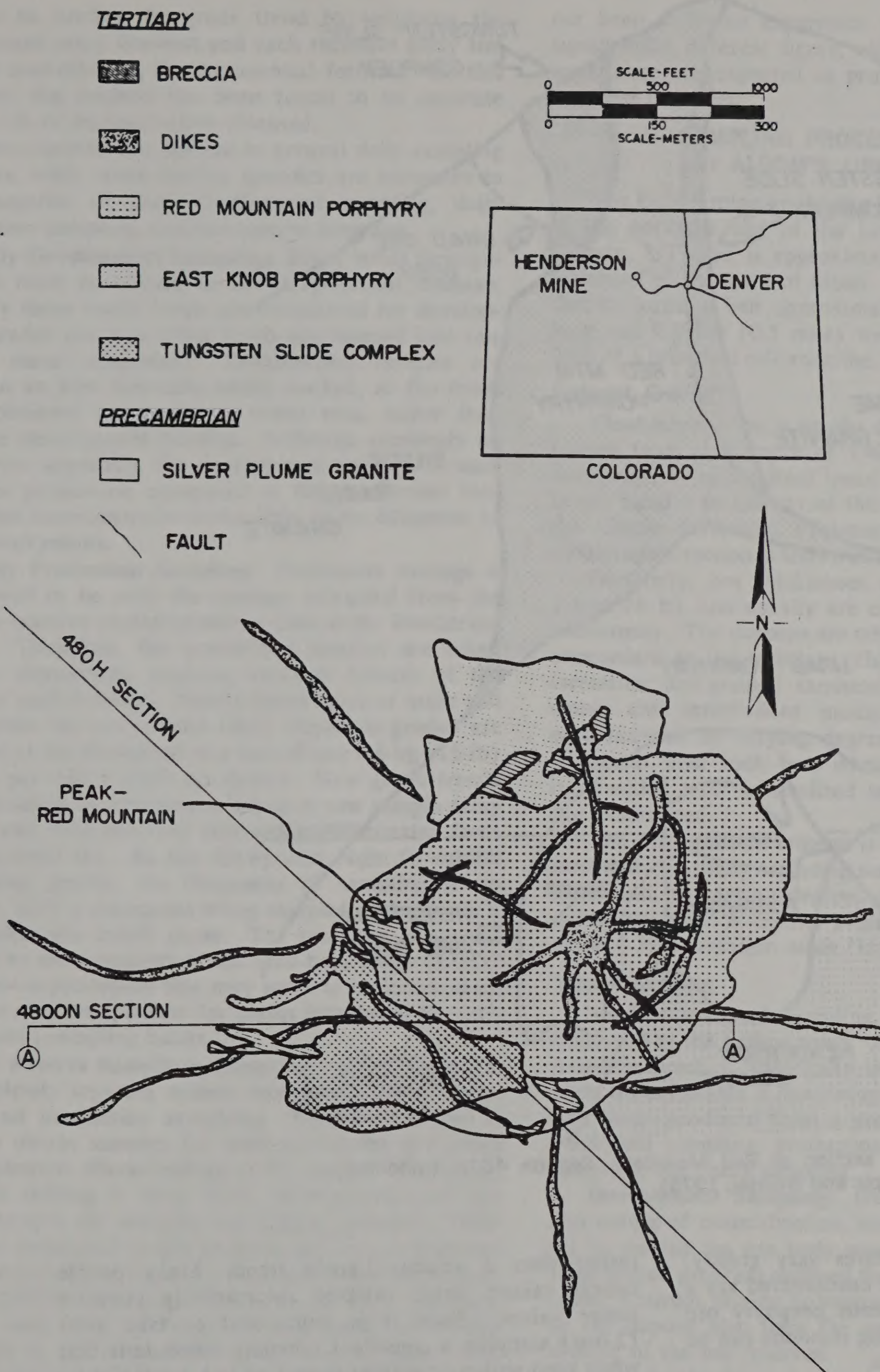


Fig. 13. Generalized geologic map of Red Mountain showing Section 480H of Fig. 14 (modified from Epis and Weimer, 1976).

drill holes, using an east-west fan pattern originating on the 8035 level. These fans are spaced roughly 61 m (200 ft) apart in the north-south direction and are designed to intercept the 0.2-0.3 zone interface at a spacing of about 61 m (200 ft). The cores obtained are logged and assayed every 3 m (10 ft), and the data are plotted on cross sections. The geology department then contours the grade zones on the cross sections according to simple averaging techniques and personal expertise.

Mine Sampling

Although every mine would like to have an absolute value for the grade being produced in each mine location, general sampling theories are not applicable in all

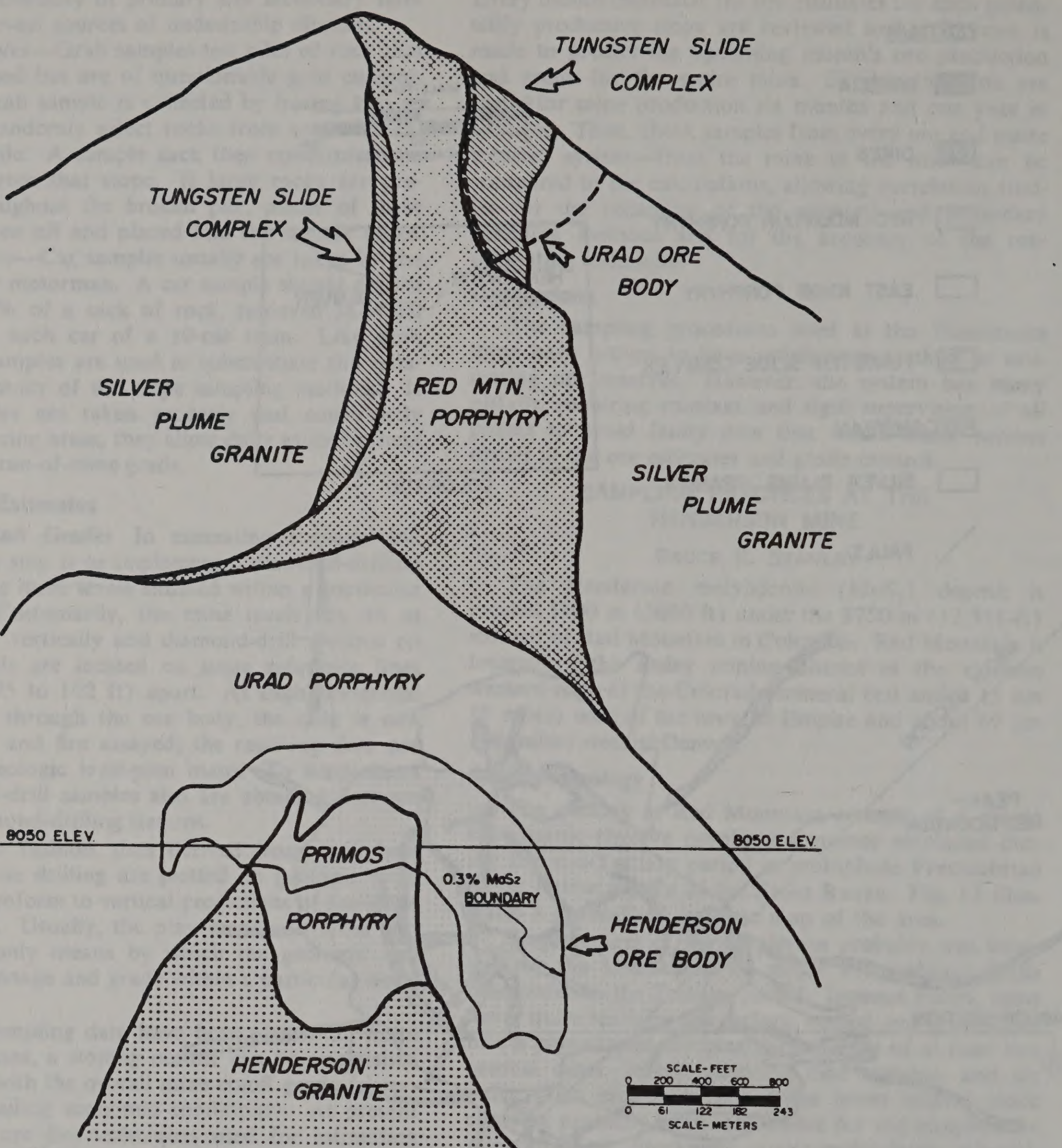


Fig. 14. Generalized geologic cross section of Red Mountain, Section 480H (modified from Epis and Weimer, 1976).

situations since mineralogical structures vary greatly. The two best situations that could be encountered are a wide vein deposit and an homogeneous porphyry ore body. For each, fairly simple sampling methods can be utilized to give reasonably good results.

Sampling in a porphyry is not a precise science. Chip and channel sampling can be difficult in many locations, leaving grab sampling the only logical alternative. Such grab sampling consists of taking pieces of rock at random from a desired location. Since a single rock cannot be expected to represent the location's grade, a 4.5 to 6.8-kg (10 to 15-lb) composite is compiled as a sample from randomly chosen pieces taken from all parts of the rock pile.

When the rock is shot or caved, it loses the structure that might have been definable when it was in place. A mathematical-trend grade then must be established, rather than a structural-grade trend. Many people believe taking larger samples automatically provides better values. Since it is impractical to take 1.8-t (2.0-st) samples, a consistent sampling schedule is best when used with a consistent sample size of 4.5 to 6.8 kg (10 to 15 lb); small individual samples produce erratic results in that type of ore body. Typically, a 1.0-g (0.03-oz) grain of 6.4-mm (0.25-in.) molybdenite contributes 0.19% to the assay of a 0.5-kg (1.0-lb) sample of 6.4-mm (1.0-in.) material.

In development and production sampling, a series of actual assays is analyzed for a "best-fit" trend, using an interactive forecasting system of computer programs. A number of methods were tested on the data, and an exponential smoothing method of forecasting was found to provide the best indication of grade trends. Using the last eight assays for any one working area, it is

possible to predict the grade trend by weighting the most recent assay heaviest and each recessive assay less heavily according to the exponential formula. At this property, the method has been found to be accurate within 5% of the true values obtained.

These theories are applied to general daily sampling practices, while some further specifics are necessary in the categories of daily development sampling, daily production sampling, and ore-reserve sampling.

Daily Development Sampling: Many levels throughout the mine contribute to the development tonnage, but only three major levels are considered for development grades (various other levels are lumped into one of the three categories). Development samples are taken in an area currently being worked, so the trend grade obtained represents an entire area, rather than any one development heading. Although seemingly an inaccurate approach, the development tonnage is such a minor proportion compared to the production tonnage that inconsistencies make little or no difference to the overall results.

Daily Production Sampling: Production tonnage is considered to be only the tonnage extracted from the cave (a massive caving system is used at the Henderson mine). Therefore, the production samples are taken only at drawpoints, assuring accurate records of the grade at each location. Newly caved areas or areas still well within the ore column (with respect to grade) are sampled at the drawpoint at a rate of one 6.8-kg (15-lb) sample per 360 t (400 st) drawn. New grade trends are calculated by computer after each new sample assay is received, with the least recent assay eliminated from the historical list. As the drawpoints begin to exhibit decreasing grades, the frequency of sampling is increased, with a drawpoint being sampled each day as it approaches the cutoff grade. The sampling is accomplished by the operators of the load-haul-dump (LHD) vehicles—a procedure that may serve to be inconsistent but also may compensate for biases introduced by each individual's sampling habits.

Ore Reserve Sampling: Sampling for ore reserves is a completely separate system from the daily development and production samplings. Diamond drilling is used to obtain samples for determining the ore zones and geological characteristics of the deposit. Currently, all such drilling is done from underground, and the cores are split for assaying and logging purposes. Drillings are conducted in fan patterns laid out to intercept a theoretical 0.2% to 0.1% interface at approximately 61-m (200-ft) intervals. The fans are lined up from east to west, with the rows of fans spaced on centers about 61 m (200 ft) apart from north to south. The drill cores are sampled in 3-m (10-ft) lengths, and the assays are entered into a computer file for easy retrieval of the associated geologic and assay information.

Conclusions

The subjects of ore reserves, grade calculations, and tonnage calculations are direct applications of geostatistics beyond the scope of this section. The calculations for percentage recovery and percentage dilution cannot be explained adequately at this time. The figures used at the Henderson mine have been taken directly from the Climax and Urad mines and from the experience derived at those mines. At Henderson, there simply has not been sufficient experience and data to develop a significantly different figure, although future data and experience are expected to provide better information for these purposes.

SAMPLING PROCEDURES AT RIO ALGOM'S LISBON MINE

The Lisbon mine, in the Big Indian district, is located at the northern end of the Lisbon Valley, San Juan County, UT, and is approximately 56 km (35 miles) southeast of the town of Moab. The Big Indian mining district forms a belt approximately 25 km (15 miles) long and 0.8 km (0.5 mile) wide on the southwestern limb of a breached salt anticline.

General Geology

The Lisbon mine is on the downthrown side of the Lisbon fault at a depth of 790 m (2600 ft) along a northwesterly mineralized trend. Mineralization occurs in the basal 9 m (30 ft) of the Moss Back member of the Chinle formation (Triassic age). A generalized stratigraphic section is shown in Fig. 15.

Generally, ore thicknesses in the district average 1.8 m (6 ft) and usually are close to the Triassic unconformity. The deposits are tabular, with the ore being concordant to the bedding. The Moss Back is fluvial-lacustrine fine-grained sandstone, conglomeritic sandstone, and interbedded mudstones. Most units are carbonaceous to varying degrees. Sandstones are the important host rock but, where mudstone lenses are interbedded with mineralized sandstones, they usually are mineralized.

The principal ore mineral is uraninite, occurring interstitially, partially replacing sand grains and mudstone clasts and, to varying degrees, also replacing carbonaceous material. A district average places the grade of ore at 0.39% uranium oxide (U_3O_8) (Wood, 1967).

Mine Sampling

Various accepted sampling procedures have been employed at the Lisbon mine. As a result, the standard sampling practice historically used in the district with slight modifications is now being used. Workable results have been produced from a program consisting of development sampling, production sampling, ore reserve sampling, and check sampling.

Development Sampling: Due to the mine's depth and nature of mineralization, close-spaced surface drilling to outline the ore body was impractical. A rather wide-spaced program outlined the major mineralized ore trend. Current mining economics and the tabular form of the deposit also limit the underground exploration drilling of the ore reserves.

A workable alternative to closed-space surface drilling is to use existing surface drill holes as a guide for mining, while keeping close control of individual mining faces on a daily basis to delimit the assay wall boundaries. Development sampling provides data enabling the mining department to balance the development ore with the stoping ore for optimum mill feed. Three times a day, every working face in the mine is scanned radiometrically. Ore and waste boundaries are marked directly on the working face. Samplers then can control mining heights for excessive overbreak or, where the ore thickness exceeds the face height, they can be adjusted accordingly.

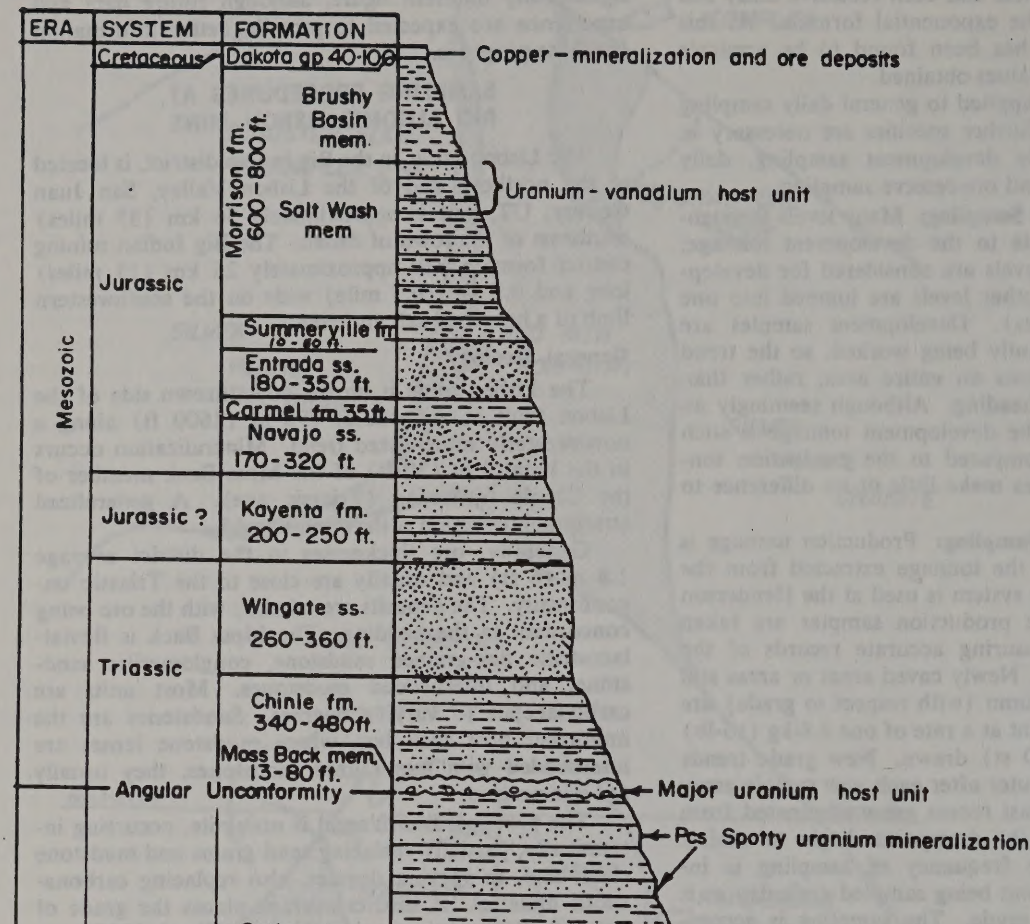


Fig. 15. Generalized stratigraphic section in the Lisbon Valley area (Wood, 1967).

When a face has not been cleaned from the previous round, the muck pile is probed for its average grade. Close sampling of this type enables better control over the mining dilution and provides grade control at the mine face by separating ore from waste before haulage.

The voluminous samples act as an averaging tool for erratic ore or any nugget effects. Since uranium mineralization is rather spotty, placing a high degree of confidence in a few samples is misleading. Daily averages of 500 to 1000 samples, along with check samples, theoretically provide a means of good data averaging for the erratic nature of the mineralization. Internal sampling errors, problems with the sampling technique, or equipment failures also become easier to detect.

As a matter of practice, chip samples at the face and grab samples of muck piles are taken on a regular basis. Since radiometric sampling is of questionable accuracy, other sampling methods are used as internal checks. Any malfunctioning or miscalibrated equipment then can be corrected. Furthermore, rock samples can be assayed for any disequilibrium inherent in uranium deposits.

In addition to scanning faces and probing muck piles, test holes are drilled at regular intervals in the development headings. The test holes generally are drilled on 6.1-m (20-ft) centers, at an angle of ±1.57 rad (±90°) and between depths of 1.8 and 6.1 m (6 and 20 ft). As many as three separate ore bands have been mined; on a regular basis, test holing defined this irregular mineralization above or below the main ore zone, as shown in Fig. 16.

Production Sampling: Production sampling simply consists of car sampling. Since a reliable daily production grade is needed for the various stopes, each ore car is probed radiometrically several times before being skipped to the surface. These car probes then are averaged by the day and computed to arrive at a daily estimate of grade and tonnage.

Although this technique is only as accurate as the individual doing the sampling, the results over an extended period usually correlate well with the ore reserve grades and mill heads. Periodic checks are performed to determine the accuracy of the probing, taking grab samples in equal proportion from each car in a train. Assays from the laboratory then are compared to radiometric probing results to detect any irregularities.

Ore Reserve Sampling: Once the pillars have been blocked out for mining, their grades and tonnages must be calculated for the mine's ore reserves. The method is similar to development sampling, with each pillar radiometrically scanned on 3-m (10-ft) centers around its perimeter. A weighted average of height vs. grade then is computed for each sample point. In turn, each weighted sample is weighted against all sampled points for the pillar. The result is the weighted height and grade for the pillar (refer to Fig. 7). From these values, it is easy to arrive at the extractable tons and kilograms (pounds) per pillar. Periodically, the mining grades from the pillars do not compare with the ore reserves exactly; the usual causes are ore variations within the pillars and, to a lesser extent, uncontrollable dilution.

Check sampling of the ore reserves is accomplished best by comparison with the production rates. Once the stoping operations begin, if excess dilution is held to a minimum as outlined by pillar scanning, production grades for the pillar life should correlate with the ore reserves. Excess dilution is easy to detect from the increased tonnage and resultant decrease in grade.

Check Sampling: Although check samples have been described in the preceding paragraphs, their importance warrants a further comment. All phases of the sampling program are checked independently, examining the efficiency of each sampling area, development sampling, production sampling, and ore reserve sampling. Thus, any improvements or corrections can be made without affecting the entire program. If all phases check with the daily sampling procedures, the entire program can be evaluated on the weighted average of each, enabling all aspects of the sampling program to be checked, adding to the validity of the results.

Recovery and Dilution

Initially, recovery estimates are an engineering problem. Factors such as the mining method, rock types,

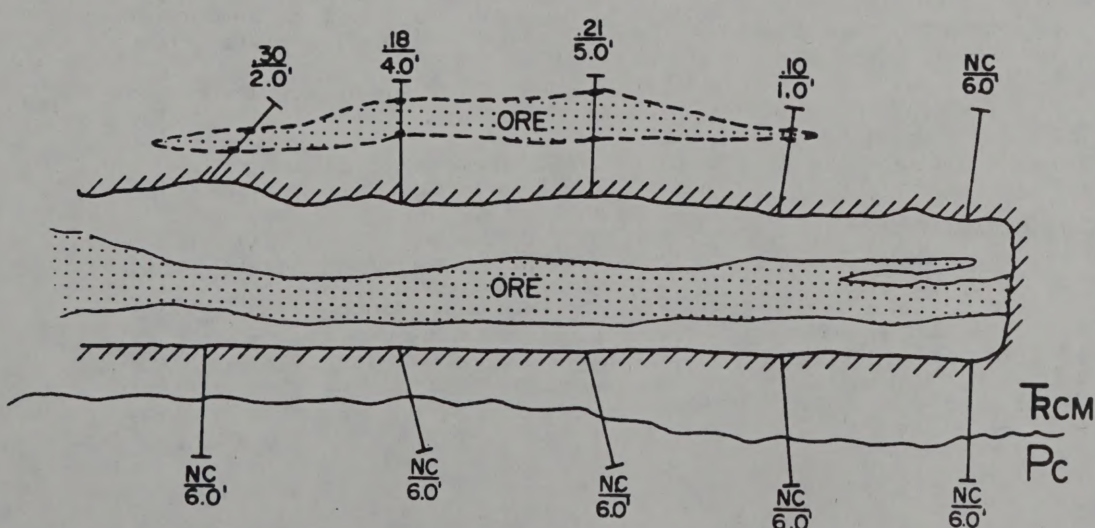


Fig. 16. Example of test holing to locate small irregular ore pods above or below the main ore zone.

Table 1. In-Place Ore Reserve Calculations

Hole Number	Area, m^2 (sq ft)	True Thickness, m (ft)	Grade, % U_3O_8	Tonnage, t (st)	Yield kg (lb) U_3O_8
52	26 268 (282,743)	1.3 (4.2)	0.21	86 184 (95,002)	180 987 (399,007)
53	11 675 (125,664)	1.7 (5.6)	0.18	51 071 (56,297)	91 930 (202,670)
54	11 675 (125,664)	0.9 (3.1)	0.29	28 272 (31,165)	81 989 (180,755)
55	8 602 (92,589)	2.0 (6.5)	0.15	43 677 (48,146)	65 516 (144,439)
Totals:	**58 219 (626,660)**	**1.4 (4.6)**	**0.20**	**209 206 (230,610)**	**420 422 (926,871)**
Adjust to 2.0-m (6.5-ft) mining height:		2.0 (6.5)	0.14	295 618 (325,863)	420 422 (926,871)
Mine Extraction at 75%:		2.0 (6.5)	0.14	221 713 (244,397)	315 316 (695,153)
Dilution at 15%:		2.0 (6.5)	0.12	254 970 (281,056)	315 316 (695,153)
Totals:		**2.0 (6.5)**	**0.12**	**254 970 (281,056)**	**315 316 (695,153)**

mine depth, and past performance are all considerations when performing the initial calculations of the mine production. Weighing all of these factors, the Lisbon mine was set up at 75% extraction and 15% dilution. Using a theoretical example of a drill intercept with 0.9 m (3.1 ft) at 0.29% U_3O_8, Table 1 lists the extraction and dilution applied to hypothetical ore reserves from surface drilling.

During the mining operations, recovery is calculated from proven ore reserves by subtracting the actual production tonnages. They are daily recovery guides and are used to modify mining operations on a daily basis. An ore reserve reconciliation is calculated every six months; partially mined pillars are planimetered and the ore reserves are revised accordingly. Any newly developed areas then are added to the reserves. As the mining operations are modified and become more efficient, an improvement over the original recovery estimate is developed.

At the Lisbon mine, dilution is reported to management in terms of mining dilution (refer to Fig. 8). In effect, this is a measurement of the mining efficiency.

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Cost Estimation for
Small Placer Mines
USBM IC 9170

Information Circular 9170

Cost Estimation Handbook for Small Placer Mines

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Donald Paul Hodel, Secretary

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CONTENTS

ILLUSTRATIONS

TABLES

UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

BCY	bank cubic yard
d/a	day per year
ft	foot
ft^2	square foot
ft^2/yd^3	square foot per cubic yard
ft/h	foot per hour
gpm	gallon per minute
h	hour
h/shift	hour per shift
hp	horsepower
in	inch
kW	kilowatt
kW/yd^3	kilowatt per cubic yard
lb/LCY	pound per loose cubic yard
lb/yd	pound per yard
lb/yd^3	pound per cubic yard
LCY	loose cubic yard
LCY/a	loose cubic yard per year
LCY/h	loose cubic yard per hour
µm	micrometer
min	minute
min/h	minute per hour
st	short ton
st/h	short ton per hour
tr oz	troy ounce
$tr\ oz/yd^3$	troy ounce per cubic yard
wt %	weight percent
yd/h	yard per hour
yd^3	cubic yard
yd^3/d	cubic yard per day
yd^3/ft^2	cubic yard per square foot
yd^3/h	cubic yard per hour
yr	year

COST ESTIMATION HANDBOOK FOR SMALL PLACER MINES

By Scott A. Stebbins[1]

ABSTRACT

This Bureau of Mines publication presents a method for estimating capital and operating costs associated with the exploration, mining, and processing of placer deposits. To ensure representative cost estimates, operational parameters for placering equipment and basic principles of placer mining techniques are detailed.

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INTRODUCTION

In 1974, the Bureau of Mines began a systematic assessment of U.S. mineral supplies under its Minerals Availability Program (MAP). To aid in this program, a technique was developed to estimate capital and operating costs associated with various mining methods. This technique, developed under a Bureau contract by STRAAM Engineers, Inc., was completed in 1975, then updated in 1983. During the course of the update, it was noted that few provisions were made for estimating the costs of small-scale mining and milling methods typically associated with placer mining. The popularity and widespread use of placer mining methods indicated that a cost estimating system for placer mining would be of value to prospectors, miners, investors, and government evaluators.

This report has been written to aid those involved with placer mining in the estimation of costs to recover valuable minerals from placer deposits. It relies on the principle that cost estimates will be representative only if calculated for technically feasible mining operations. Because the design of such an operation can be difficult, provisions have been made to assist the user in achieving this goal.

Section 1 of the report describes the processes involved in placering, and may be used to aid in designing a viable mine. Operational parameters for equipment commonly used in placer exploration, mining, and processing are discussed, as well as basic principles of successful placer mining techniques. If the reader is unfamiliar with this form of mining, section 1 should be thoroughly understood prior to estimating costs.

Section 2 contains cost equations that enable the user to estimate capital and operating costs of specific placer techniques. Cost equations are designed to handle the wide variety of conditions commonly found in placer deposits. This allows the reader to tailor estimates to the characteristics of a particular deposit, which ensures representative costs. Although based primarily on gold placer operations, cost equations are valid for any other commodity found in deposits of unconsolidated material. Equations are geared to operations handling between 20 and 500 LCY/h of material (pay gravel plus overburden). Estimated costs are representative of operations in the western United States and Alaska, and are based on a cost date of January 1985.

The appendix provides an example of placer mine design and cost estimation using the information contained in this report.

This report is not intended to be an exhaustive discussion of placer mining. Many detailed texts have been written on this process, any one of which will assist the reader in method design. A number of these are listed in the bibliographies accompanying sections 1 and 2.

ACKNOWLEDGMENTS

A special debt is owed to the late George D. Gale, metallurgist, Bureau of Mines. This handbook, and many of the ideas and facts it contains, are the product of his ingenuity.

SECTION 1.—PLACER MINE DESIGN

The complete design of a placer mine involves the integration of exploration, mining, processing, and supplemental systems for the efficient recovery of valuable minerals from an alluvial deposit. This design is the first step in accurate cost estimation.

In this section, individual systems are categorized as follows:

1. Exploration.—The phase of the operation in which resources are delineated. Because the amount of time and effort spent on discovery is difficult to tie to any one specific deposit, only the processes of delineation and definition are costed. Field reconnaissance, drilling, and panning are representative of items in this category.
2. Mining.—Deposit development, material excavation and transportation, and feeding of the mill are all included in this category. Items such as clearing and overburden removal are also included.
3. Processing.—Processing is defined as all tasks required to separate the desired mineral products from valueless material.
4. Supplemental.—Any items not directly related to mineral recovery, but necessary for the operation of the mine. These might include buildings, employee housing, and settling pond construction.

Before designing a placer mining operation, the evaluator will need information concerning the deposit under evaluation. Preliminary information helpful in exploration program, mine, mill, and supplemental function design includes

1. Description of deposit access.
2. Anticipated exploration and deposit definition requirements.
3. An estimate of deposit geometry and volume.
4. Distribution and location of valuable minerals within the deposit.
5. Geologic characteristics, volume and depth of overburden.
6. Depth, profile, and geologic characteristics of bedrock.
7. Local topography.
8. Physical characteristics and geologic nature of valuable minerals.
9. Availability of water.
10. Availability of power.
11. Environmental considerations.
12. Labor availability and local wage scales.
13. Housing or camp requirements.

Information should be as detailed as possible. By providing such items as exact haul distances and gradients, accurate estimates of overburden thickness and deposit area, the evaluator will increase the precision of cost calculations.

With the preceding information in hand and the help of the material contained in the following pages, the user will be able to design a technically feasible operation. The following sections will assist the evaluator in planning each phase of the mine. When designing systems for individual areas of operation, the evaluator must keep in mind that these systems will interact and must be compatible. For instance, hourly capacity of pay gravel excavation should equal mill feed rate, and the mill must be set up to easily accept gravel from the equipment used for material transportation.

Most of the information contained in the following pages is based on average operating parameters and performance data for the various types of equipment used in placer mining. Costs and conclusions derived from this manual must be considered estimates only. Because of the many variables peculiar to individual deposits, the stated levels of equipment performance and costs may not be realized on any given job.

EXPLORATION

It can be safely stated that far more people seek placer deposits than actually mine them. Exploration for placer gold can be enjoyable work and has achieved a recreational status in the western United States. For the serious miner, however, exploration is only the initial phase of a complete mining operation. Consequently, it incurs a cost that must be repaid by the recovery of valuable minerals.

For the purposes of this report, exploration is divided into two phases. The first phase involves locating the deposit, and the second consists of defining enough of a resource to either justify development or to eliminate the deposit from further consideration.

Costs for the first phase of exploration are difficult to attribute to any one deposit. This type of exploration is typically regional in nature and deposit specifics are rarely considered. For cost estimation purposes, expenses associated with a specific deposit are the main concern. Only costs directly related to the definition of that particular deposit will be calculated. Accordingly, this discussion deals mainly with the deposit definition phase of exploration.

Time, effort, and money spent on resource definition vary greatly from one deposit to the next. Some miners are satisfied with the degree of certainty obtainable with shovel, pan, and physical labor. Others, wishing more security, systematically trench or drill the deposit and process samples using some sort of mechanical concentrator. Still others, hoping for greater assurance, follow up drilling or trenching by bulk sampling using machinery intended for mining. These samples are then processed in a scaled-down version of the proposed mill. The extent of effort spent on deposit definition is related to

1. Degree of certainty desired.
2. Availability of capital.
3. Experience of the operator.
4. Historical continuity of similar or local deposits.

It is intuitively obvious that the degree of certainty of success is related to the extent of exploration undertaken, and it is desirable to delineate the deposit as extensively as is practical prior to production. In many cases, however, lack of exploration capital and the need for cash-flow limit

the exploration phase, and mining commences on the limited information at hand. Goals of a thorough exploration program include determination of

1. Deposit volume.
2. Deposit and overburden geometry.
3. Deposit grade.
4. Distribution of valuable minerals within the deposit.
5. Geological and physical characteristics of the valuable minerals.
6. Geological and physical characteristics of waste material.
7. Location, geology, and physical nature of the bedrock.
8. Water availability.
9. Environmental concerns.

Much of the information needed to estimate costs of developing and operating a placer mine is gathered during deposit exploration. Consequently, costs estimated after exploration are much more precise than estimates made prior to exploration.

In section 2 of this report, two methods are presented for estimating exploration costs. With the first, a cost can be calculated by simply estimating the total resource of the deposit. This method is based on total exploration expenditures for several active placer operations, but is not considered as precise as the second method.

The second method requires that the evaluator design an exploration plan. This plan should include the type and extent of each exploration method required, for example

1. General reconnaissance, 5 days with a two-person crew.
2. Seismic surveying, 10,000 linear ft.
3. Churn drilling, 4,000 ft.
4. Trenching, 1,000 yd^3.
5. Samples panned, 2,000.
6. Camp facilities, four people for 20 days.

To aid in developing this plan, some techniques commonly employed for sampling and subsurface testing of placer deposits are discussed in the following paragraphs. These include panning, churn drilling, bucket drilling, rotary drilling, trenching, and seismic surveying.

PANNING

One of the most versatile and common sampling devices in placer mining is the gold pan. It is used as a reconnaissance tool, a sampling tool, and a concentrate refining tool. With a gold pan, the prospector has the ability to, in effect, conduct his or her assay work on-site with immediate results. Although accuracy may be poor, the prospector can determine in the field if gold is present and in roughly what amounts.

The gold pan uses gravity separation to concentrate heavy minerals. Pans come in a variety of sizes, ranging in diameter from 12 to 16 in. An experienced panner can concentrate approximately 0.5 yd^3 gravel daily. Because of this limited capacity, panning can be costly when large volumes must be processed; however, low capital expense, ease of use, versatility, and portability make the gold pan invaluable.

Immediate feedback when exploring or mining is a prime advantage of the gold pan. This one feature is extremely important for eliminating areas of low potential during exploration, and for separating pay gravel from waste during production. Skilled use of a gold pan during the mining sequence can make or break the small mining operation.

CHURN DRILLING

Methods of drilling placer deposits are quite varied, but the most common technique is churn drilling. Typically, the churn drill uses percussion to drive casing down through the material being sampled (in some instances, casing is not used). After a length of casing is driven, the contents are recovered (bailed), another length of casing is added, and the process is repeated. Depths are usually restricted to less than 150 ft, and hole diameters range from 4 to 10 in.

One advantage of this method is that sample processing keeps pace with drilling, allowing good control of drill-hole depth and instantaneous logging. A churn drill is generally operated by two people; the driller operates the drill, bails the sample, and keeps track of the depth of each run; the panner estimates the volume of the samples, pans them as they are recovered, and logs the hole.

Drilling rates average about 2 ft/h but can reach as much as 4 ft/h in clay, soil, sand, pebbles and soft bedrock. The machine is suitable for drilling through cemented gravels and permafrost, although productivity will diminish. Penetration is drastically reduced in ground containing boulders and in competent or hard bedrock.

Samples recovered from churn drill casings are often subject to volume changes caused by compaction or expansion of material within the casing. Sample volume changes can also be caused by compaction around the bit forcing material out into the surrounding formation, and by material "run-in" due to high deposit water content. One or more of these conditions may be encountered in any one deposit, requiring the application of volume corrections. This task is often difficult and requires the experience of a qualified driller or engineer.

BUCKET DRILLING

Bucket drilling, although not as popular as churn drilling, has important applications in placer deposit evaluation. Under ideal conditions, this technique is relatively fast and provides large samples. In this system, a standard rotary drill is equipped with a special "bucket" bit consisting of a 30- to 48-in-diam cylinder, 3 to 4 ft long. The bit is driven down through the deposit, using the rotational force of the drill, until the cylinder is full. As the bit is withdrawn, a mechanism closes off the bottom of the bit retaining the sample. The process is then repeated until the desired depth is reached.

Bucket drills perform best in sands, soils, pebbles, and clays. Progress is slow, and sometimes impossible, in ground containing boulders, cemented gravel layers, and bedrock. The size of the bit tends to disperse drilling force over a large area, thereby reducing the effective penetration rate. For this reason the bucket drill quickly becomes inefficient in hard or compact material. Problems are also encountered in saturated ground, where water often washes away a portion of the sample as the bit is withdrawn.

Bucket drilling extracts a much larger sample than other drilling methods. Consequently, the influence of the bit on compaction and expansion of material is reduced.

ROTARY DRILLING

This type of drill, commonly used for drilling large-diameter blastholes in surface mining, has found limited use in placer exploration. The only way to obtain a sample with this machine is to analyze drill cuttings. Because the method does not provide a core, it is difficult to associate a volume with the recovered material, and it is hard to estimate the depth horizon of the sample.

Rotary drills are useful in that they provide a fast, inexpensive way to determine the depth of bedrock. Holes provided by rotary drills range from 6 to 15 in. in diameter and reach any depth required for placer mining. Virtually any material can be drilled, and penetration rates are far superior to any other placer drilling method. Regardless of the steps taken, however, it is difficult to accurately estimate deposit grade with samples obtained from rotary drilling.

TRENCHING

In fairly shallow, dry deposits, trenching with a backhoe is an extremely effective sampling technique. The procedure involves digging a trench to bedrock, then obtaining material from a channel taken down one side of the trench. This material is then measured and analyzed, providing a grade estimate. Another method relates an assay analysis of all the material extracted by the backhoe to the volume of the trench. The disadvantage of this method is the inability to determine the horizon of valuable mineral concentration. With either method, large-volume samples are available at a low cost.

In sampling situations, backhoes can excavate from 20 to 45 LCY/h. Sample control is typically good with little volume distortion or material dilution under properly controlled circumstances. Backhoes are relatively inexpensive, easy to operate, versatile, and readily available. The machine can dig a variety of formations, and digging depths as much 30 ft below the machine platforms are possible. In saturated ground, keeping the trench open for sampling is normally a major problem.

SEISMIC SURVEYS

In placer mining, bedrock depth plays a key role. Although not always the case, gold tends to concentrate near, on, or even in bedrock in a majority of placer deposits. Consequently, it is imperative to understand the nature of the bedrock and to design a mining method and select equipment based on its depth.

One method of determining bedrock depth is seismic refraction or reflection. In simple terms, the technique involves bouncing sound energy off the relatively resistant bedrock to determine its depth. The method is much cheaper than drilling a series of holes and, if bedrock proves to be too deep for practical mining, may prevent unnecessary drilling.

MINING

Next, a method for excavation and and transportation of material contained in the deposit is needed. Mining methods are typically dictated by several basic factors. Deposit depth, size, and topography are of primary importance. The geologic nature of the deposit and accompanying overburden both play key roles. Types of equipment obtainable locally, sources of power, and the availability of water are all important factors. In some cases, operators may simply feel more confident using one method of extraction as opposed to another, even if local conditions are unfavorable.

In any event, the mining method should be designed with one fundamental goal in mind: To extract pay gravel from the deposit and move it to the mill at the lowest possible overall cost. Several basic concepts should be designed into the mining method to keep costs low. These include

1. Haul only pay gravel to the mill. Eliminate hauling and processing unprofitable material.
2. Handle both overburden and pay gravel as few times as possible. Do not pile overburden or tails on ground that is scheduled for excavation.
3. Locate the mill at a site that minimizes average pay gravel haul distance. In most instances, it is cheaper to pump water than to haul gravel.
4. Do not mine gravel that is not profitable even if it contains gold. Money is lost for every yard of gravel mined if that gravel does not contain enough value to pay for the cost of mining and processing.

As can be seen, common sense plays a large role in the proper design of a placer mine. The same holds true for mine equipment selection. Countless combinations of equipment have been tried in attempts to effectively mine placer deposits. Equipment typically used in the western United States includes

1. Backhoes (hydraulic excavators).
2. Bulldozers.
3. Draglines.
4. Dredges.
5. Front-end loaders.
6. Rear-dump trucks.
7. Scrapers.

Each type of equipment is suited to a particular task. In some instances, only one piece of equipment may be used to remove overburden, excavate and haul pay gravel, and place mill tailings and oversize (i.e., bulldozers). More often, several different types of equipment are utilized to take advantage of their specific attributes.

When selecting placer mining equipment, the evaluator must consider two important concepts. First, the volume of earth in place is less than the volume of the same earth after excavation. This point is critical in cost estimation and must be remembered. Because placer gravel is relatively light, placer mining equipment is typically limited by volume capacity, not weight capacity. For this reason, mine equipment capacities and associated cost equations in this report are based on volume *after* accounting for material swell—in loose cubic yards. Resource estimates are typically stated in bank cubic yards—the volume *before* accounting for material swell. This has a significant meaning to the design of a placer mining system. To mine a 500,000-BCY

deposit, equipment will have to move 570,000 LCY of gravel if the material swells 14% (typical for gravel deposits). Although the total weight of material moved is constant, equipment will have to move a larger volume of gravel than the in-place estimate indicates. As a result, the mining system should be designed around the total loose cubic yards of gravel to be moved, not the total bank cubic yards.

Second, mine equipment equations in section 2 of this report are based on the *maximum* amount of overburden, pay gravel, and mill tails moved daily. Although average volume handled might be less, equipment must be selected to handle the maximum load.

To aid in mine planning, and to obtain reasonable capital and operating mine costs, the following information will typically be required:

1. Total length and average width of haul and access roads.
2. Total surface area of deposit.
3. Nature of ground cover.
4. Topography of deposit area.
5. Total loose cubic yards of overburden, and maximum amount of overburden handled daily.
6. Total loose cubic yards of pay gravel, and maximum amount of pay gravel handled daily.
7. Total cubic yards of mill tails handled daily.
8. Type of equipment desired.
9. Average haul distances and gradients for overburden, pay gravel, and tailings.

The following is a discussion of the principal types of equipment used in excavating and hauling overburden, placer gravel, and mill oversize and tails, and may be used to aid in mine design and equipment selection.

BACKHOES (HYDRAULIC EXCAVATORS)

The backhoe is one of the most efficient types of equipment for bedrock cleanup. It is most often used for the extraction of pay gravel, but can also be used for excavation of overburden. The machine has almost no capacity for transportation of material and for that reason is used in conjunction with either front-end loaders, trucks, or in some cases, bulldozers. Depending on bucket selection, the machine can handle a variety of ground conditions including clays, poorly sorted gravels, tree roots, and vegetation. Digging depths of over 30 ft are obtainable with certain backhoes, but production capability decreases rapidly as maximum digging depth is approached.

Backhoes typically used in the western United States are capable of excavating from 95 to 475 LCY/h. Sizes range from 105-hp machines with 0.5-yd^3 buckets to 325-hp units with 3.75-yd^3 buckets. Capacity is contingent upon digging difficulty, operator ability, swing angle, digging depth, and obstructions.

The backhoe is ideal for situations where bedrock cleanup is critical, obstructions exist in the mining area, and other means of transporting gravel are available.

BULLDOZERS

The bulldozer represents an extremely versatile tool in placer deposit extraction, and is the most popular. It can be used for overburden removal, pay gravel excavation, bedrock cleanup, overburden and pay gravel transportation, road construction, tailings placement, and a variety of minor functions. The bulldozer is the only device capable of handling all tasks required for placer mining in a practical manner and must be considered if capital is scarce.

Although bulldozers can handle all placer mining functions, they are not necessarily the most efficient machine for any one task. With its ripping capacity, the bulldozer is capable of cleaning up bedrock; however, the backhoe is much more selective and efficient. The bulldozer can, and often is, used to transport gravel, but in most cases trucks, scrapers, and front-end loaders can each do the job cheaper if haul distances are more than a few hundred feet. In addition, bulldozers are not well suited to more large volumes of gravel or to dig to excessive depths. In both instances, draglines exhibit superior performance.

A major advantage of the bulldozer is its ability to excavate, transport, and load the mill all in one cycle, eliminating the need for expensive rehandling. Dozer capacities for excavating and hauling range from 19 LCY/h for a 65-hp machine up to 497.5 LCY/h for a 700-hp dozer (based on a 300-ft haul distance). Capacity is dependent upon ripping requirements, operator ability, cutting distance, haul distance, digging difficulty, and haul gradient.

Dozers are best suited for situations where deposit and overburden thicknesses are not excessive, few large obstructions are present, and haul distances average less than 500 ft.

DRAGLINES

Draglines are well suited for excavating large quantities of overburden, gravel, and waste. Although their material transporting ability is limited, draglines with booms up to 70 ft long are capable of acting as the sole piece of mining equipment. As with the bulldozer, draglines can excavate overburden and pay gravel, load the mill, and remove tailings; however, draglines are relatively inefficient at bedrock cleanup, and do not handle difficult digging as well as backhoes or dozers.

Depths of over 200 ft are obtainable with this type of machine, and when used in conjunction with front-end loaders or rear-dump trucks, large-capacity operations are possible. Draglines handle from 28 LCY/h for a 84-hp unit to 264 LCY/h for a 540-hp machine. Capacity is dependent upon bucket efficiency, swing angle, and operator ability.

Draglines are ideal for overburden removal and for large, deep deposits where bedrock cleanup is not critical. They must, however, be matched with the right equipment (i.e., portable mills or gravel transportation machinery).

DREDGES

Cost estimation equations for dredging are not included in this report. Dredges, except for recreational units and small machines used in active channels, are designed for high-capacity excavation of specific placer environments. The machines are best utilized in large volume, relatively flat-lying deposits that occur below water level. Because of large capital investment requirements and a scarcity of ground suitable for large-scale dredging, they are uncommon in the western United States.

Operating costs for large-capacity dredges average approximately \$0.70/yd^3. Purchase and refurbishing costs are often more than \$3 million, and can run over \$10 million. In large-volume situations, dredges must be considered. Because suitable applications are rare, however, they have not been included in this report.

FRONT-END LOADERS

This versatile machine is capable of many functions. In the western United States, its primary use is hauling previously excavated gravels, and the subsequent loading of the mill. Although front-end loaders are not the most efficient hauling unit, their self-loading ability provides many advantages. One is the elimination of the need to match the excavation machine with the haul unit. With a front-end loader, the excavator can operate at its own pace and simply stockpile material. The loader then feeds from the stockpile and transports gravel to the mill feed hopper. This removes the problem of matching excavator output with truck cycles or mill feed rates.

The machine is also capable of removing and transporting mill oversize and tailings; however, front-end loaders are not particularly adept at excavating consolidated material. If overburden or gravel are at all compacted, a backhoe or bulldozer should be used for a primary excavation.

Front-end loaders are capable of hauling from 24 LCY/h for a 65-hp, 1-yd^3 machine to 348 LCY/h for a 690-hp, 12-yd^3 machine (based on a 500-ft haul distance). Capacity varies with haul length, haul gradient, operator ability, bucket efficiency, and type of loader.

Front-end loaders are best utilized as haul units over distances of less than 1,000 ft. Their versatility makes them useful for pay gravel and overburden transportation, mill oversize and tailings removal, and general site cleanup.

REAR-DUMP TRUCKS

Trucks represent the least expensive method of material movement over long distances; however, since other machinery is required for loading, total gravel transportation expenses over short distances may be higher than for front-end loaders or scrapers. Trucks generally serve two purposes: Material movement and mill feed. They have relatively low capital costs and require little maintenance compared to other placer equipment. Trucks do need fairly good road surfaces and require careful matching with loading equipment to achieve acceptable efficiency.

Capacities for units at small placer operations range from 3 to 47.5 yd^3. Trucks are most productive over haul distances of 1,000 to 10,000 ft and can travel faster than equivalent-sized scrapers or front-end loaders. Production capacities range from 32.3 LCY/h for a 3-yd^3 truck to 444.8 LCY/h for a 47.5-yd^3 truck (based on a 2,500-ft haul distance). Capacity is contingent upon loader capacity, haul distance, and haul gradient.

Trucks are suited to operations where a fixed mill is situated more than 0.5 mile from the minesite. They are equally effective hauling pay gravel, overburden, or mill tailings and oversize, but must be accompanied by a method of material loading.

SCRAPERS

These machines are noted for their high productivity when used to transport overburden, pay gravel, and tailings. As with front-end loaders, scrapers are self-loading, although bulldozers or other scrapers often assist. They are capable of much higher speeds and greater capacity than front-end loaders, and exhibit haulage characteristics similar to rear-dump trucks. Scrapers, however, are more costly to purchase and maintain.

Scrapers are limited in their ability to excavate consolidated or unsorted material. A bulldozer equipped with a ripper must precede them in overburden or gravel that is not easily drifted. If boulders are present, they must either be blasted or removed by other means. The nature of the scraper-dumping mechanism renders them unsuitable for direct mill feed. When used to haul pay gravel, scrapers will typically unload near the mill, and bulldozers will then be used to feed material.

Capacities range from 201 LCY/h for a 330-hp machine to 420 LCY/h for a 550-hp machine (based on a 1,000-ft haul distance). Capacity is contingent upon haul distance and gradient, and loading procedure.

In placer mining, scrapers are best utilized for transportation of unconsolidated overburden or mill tailings over distances ranging from 500 to 5,000 ft.

PROCESSING

Often the most difficult part of placer mining is achieving the desired recovery of valuable minerals from mine-run gravel. The design of a successful mill is a specialized science and often proves difficult even for those actively involved in placer mining. Great care must be taken to ensure the recovery of a high percentage of contained valuable minerals. Obviously, the profitability of an operation is directly related to the percentage of contained valuable minerals recovered by the mill.

Although mill design can be difficult, the basic premise used in heavy mineral recovery is quite logical. In placer deposits, high-density minerals have been concentrated by combinations of natural phenomenon such as gravity, turbulent fluid flow, and differences in mineral density. Consequently, it would seem practical to utilize these conditions to further concentrate heavy minerals. This form of mineral recovery is referred to as gravity separation and is the basis for most placer mills.

Gravity processes must consider both particle specific gravity and size for effective separation. Differences in specific gravity alone will not distinguish various materials. It is the differences in weights in a common medium that creates efficient separation. Consequently, a particle of high specific gravity and small size may react the same as a large

particle with low specific gravity in a given fluid. If gravity separation is to be effective, size control must be implemented to take advantage of differences in particle specific gravity.

Equipment used for gravity separation ranges from gold pans to prebuilt self-contained placer plants. In general, the most widely employed devices in the western United States are

1. Jig concentrators.
2. Sluices.
3. Spiral concentrators.
4. Table concentrators.
5. Trommels.
6. Vibrating screens.

Of these devices, trommels and vibrating screens are used for particle size classification, and the remainder are forms of gravity concentrators. In addition, feed hoppers and conveyors are needed for surge capacity and material transportation. These items, which are commonly neglected in plant costing, must be carefully selected to ensure proper plant operation.

Although the complete design of a placer recovery plant cannot be thoroughly covered in the space available here, three sample flowsheets illustrating basic placer mill design are included at the end of this section on processing. Along with a flow sheet detailing equipment type, size, and capacity required for the mill, the following will be needed to obtain an accurate cost estimate using this report:

1. Maximum feed capacity of the mill.
2. A material balance illustrating feed, concentrate, and tailings rates.
3. The purpose of each gravity separation device (rougher, cleaner, scavenger, etc.).
4. Method of removal and transportation of mill tails and oversize.

The following discussion details equipment used in gravity separation and may prove useful in mill design.

CONVEYORS

As material travels through a mill circuit, it can be moved by conveyor, pumped in a slurry, or transferred by gravity. In placer processing mills, material is most often transported in a slurry or by gravity. In some cases, however, conveyors are necessary. Conveyors are typically used for situations of extended transport where material need not be kept in a slurry, such as the removal of oversize or tailings. They provide an inexpensive method of transporting large quantities of material over fixed distances. In the case of placer processing plants, this distance typically ranges between 10 and 120 ft. Conveyors used in these plants are typically portable, and consequently come complete with framework and support system ready to operate.

Conveyor capacity is related to belt width, belt speed, and material density. For most placer gravels, capacities range between 96 yd^3/h for an 18-in-wide belt to 480 yd^3/h for a 36-in-wide belt.

FEED HOPPERS

The initial piece of equipment in most mill circuits is a feed hopper. The hopper is used in conjunction with a feeder to smooth out material flow surges introduced by loading devices with fixed bucket sizes (front-end loaders, rear-dump trucks, etc.). Hoppers often contain a grizzly in order to reject large oversize material. The feeder, typically a vibrating tray located under the hopper, transfers gravel at an even rate to the circuit. Although the hopper-feeder combination may appear to be a minor piece of equipment, a steady flow of material through the mill is very important for effective gravity separation.

Hopper capacity and feeder capacity are two separate items. Generally, hoppers are designed to hold enough material to provide a steady flow of gravel despite surges inherent in mining cycles. Feeders are set to provide the appropriate flow rate to the mill. So even though a hopper may have a 100-yd^3 capacity, the feeder might provide material at 20 yd^3/h.

Feeders are not always used in placer mills. When they are not used, feed rate is regulated by the size of the opening in the bottom of the hopper. The cost estimation curves in this report calculate hopper-feeder costs based on feeder capacity, which typically equals mill capacity. Factors are provided for situations where feeders are not used.

JIG CONCENTRATORS

Jigs are gravity separation devices that use hindered settling to extract heavy minerals from feed material. They typically consist of shallow, perforated trays through which water pulsates in a vertical motion. In most instances, a bed made up of sized shot, steel punchings, or other "ragging" material is placed over the perforations to promote directional currents required for separation. Slurried feed flowing over the bed is subjected to the vertical pulsations of water, which tend to keep lighter particles in suspension while drawing down heavier constituents. These heavy minerals are either drawn through the bed and discharged from spigots under the jig or, if too large to pass through the perforations, are drawn off near the end of the machine. Lighter particles continue across and over the end of the jig as tailings.

Jigs are sensitive to feed sizing. They are generally utilized for feeds ranging from 75 μm to a maximum of 1 in, but recoveries improve if feed is well sized and kept to minus 0.25 in. Efficiency is maximized when feed materials have been deslimed and sized into a number of separate fractions for individual treatment. Optimum solids content for jig plant feed ranges from 35% to 50%—the object being to avoid excessive dilation of the material. Capacities for jigs range from 0.1 to 400 yd^3/h and are dependent upon desired product as well as equipment size.

SLUICES

The most common gravity separation device used in placer mills, sluices are simple to construct, yet effective heavy mineral recovery tools. Sluice design is quite diverse and opinions differ widely with respect to capacity, riffle design, and recovery. In general, capacities and performances vary with box width and slope, gold particle size, nature of feed, and availability of water.

Sluices are primarily used for rough concentration and are capable of processing poorly sorted feeds. As with other methods, however, recovery is related to the degree of previous sizing.

Sluice design can be quite complex but usually is a matter of trial and error. Several basic principles typically apply. Width is determined by the maximum and minimum volume of water available, the size and quantity of oversize feed that must be transported, and the slope. Length depends principally on the character of the gold. Coarse gold and granular gold settle quickly and are easily held in the riffles, while fine gold and porous gold may be carried some distance by the current. Velocity of the water is controlled primarily by the slope. In general, the sluice should be constructed and installed so that water flowing through the box will transport oversized material and prevent sand from packing the riffles.

If the surface of the water flowing through the sluice is smooth, the bottom of the sluice is probably packed with sand, allowing little gold to be saved. The desired condition occurs when waves form on the surface of the water flowing through the sluice, and these waves, along with the wave-forming ridges of material on the bottom of the sluice, migrate upstream. This indicates an eddying or boiling activity on the lee side of the ridges, which maximizes gold recovery and tailings transport. Consequently, the sluice attains maximum efficiency when riffle overloading is incipient.

Sluices are generally considered to be high-capacity units, with a 12-in-wide sluice box capable of handling 15 yd^3/h if sufficient water is available. A 24-in-wide sluice can handle up to 40 yd^3/h, and 48-in-wide sluices have reportedly processed up to 200 yd^3/h. Of course, a sluice will handle as much gravel as the operator wants to push through it. However, to ensure reasonable recovery, capacity is limited by box width and slope, water availability, and feed characteristics.

Feed slurry densities are highly variable and range from 1% to 35% solids by weight, averaging 10%. Water use can be reduced significantly if the larger of the oversize is eliminated from the feed. Sluices require no power to operate unless a pump is needed to transport water or slurry. One disadvantage of the sluice is the necessity to halt operations in order to recover concentrates.

SPIRAL CONCENTRATORS

Spirals are used infrequently in the western United States but may be applicable for certain types of feed. These gravity separation devices exhibit several desirable features. They accept sized slurry directly, and require no energy to operate other than perhaps pumps for material feed. Pumps can be excluded if gravity feed is used. Selectivity is high because of adjustable splitters within the slurry flow. Spirals can be used to produce a bulk concentrate, scavenge valuable minerals from tailings, or in some instances, recover a finished concentrate. The ability to produce a finished concentrate will be limited to feeds that contain a higher concentration of desired product than typically found in most gold placer feeds.

To save space, two or three spiral starts are constructed around a common vertical pipe. This arrangement takes little floor space, allowing banks of multiple units to be set up for large-capacity requirements. In this situation, slurry distributors are required to sectionalize feed for individual spirals.

Maximum feed rates vary according to feed particle density, size, and shape. Rates generally range from 1.0 to 1.4 yd^3/h roughing down to 0.3 to 0.5 yd^3/h cleaning per start. Feed slurry density is typically less than 25% solids by weight, necessitating the use of larger pumps than needed for jigs or tables.

TABLE CONCENTRATORS

Concentrating tables (shaking tables) are one of the oldest methods of mechanical gravity concentration. Although capable of handling a variety of feed types and sizes, their optimum use is wet gravity cleaning of fine concentrates ranging from 15 μm to 1/8 in. The unit consists of a large, flat, smooth table, slightly tilted, with riffles attached to the surface. A longitudinal reciprocating motion is introduced to the deck by means of a vibrating mechanism or an eccentric head action.

Although limited in capacity, tables have the advantage of being easily adjustable by regulating the quantity of wash water and altering the tilt angle of the deck. The results of these changes are immediately observable on the table. With the addition of splitters, efficient control of high-grade concentrate recovery, middling recovery, and tailings production is possible.

Solids content for table feeds averages approximately 25% by weight. Stroke length and speed are adjusted according to feed. Long strokes at slow speeds are used for coarse feeds; fine material responds better to short strokes at higher speeds. A reciprocating speed of 280 to 380 strokes/min will handle most feeds. Table capacities range from 0.05 to 8 yd^3/h and depend on desired product as well as equipment size.

TROMMELS

This machine is the most common size classification device used in gold placer mills and is well suited for this task if properly designed. Trommels consist of a long rotating cylinder that is typically divided into two sections.

In the first section, lengths of angle iron or similar material are fastened to the inside of the rotating drum. These act as lifters to carry feed up the side of the rotating cylinder. As material reaches the top of the rotation, it falls back to the bottom of the cylinder and breaks upon impact. This action, along with water introduced under pressure, serves to break up compacted soils and clays, and liberate valuable minerals.

The second section consists of perforations in the cylinder walls positioned along the length of the drum. Typically, perforation size will graduate from 1/8 in, to 3/16 in, to 1/4 in as the feed progresses down the trommel.

Sized fractions are drawn directly below the section of the trommel in which they are separated. They generally flow to either a vibrating screen to be sized further or to a gravity separation device. Oversize material is discharged out the end of the trommel as waste.

Trommels are particularly well adapted to placer feeds because of their ability to handle a diversity of feed sizes and to break up material in the scrubber section. Capacity ranges from 10 to 500 yd^3/h and is dependent on feed characteristics, screen perforation sizes, and machine size. Water requirements are contingent upon the amount of washing desired.

VIBRATING SCREENS

Vibrating screens are often used for secondary size classification in circuits treating alluvial ores and, in some cases, may provide primary sizing. The machines consist of a deck, or decks, containing inclined screening surfaces that are vibrated in either a rectilinear or elliptical motion. Screening medium can be woven wire cloth, parallel bars, or punched sheet metal.

High capacity, ease of installation, and reasonable operating costs have all contributed to the popularity of vibrating screens. The practical minimum size limitation for production screens is about 100 mesh, although 325-mesh separations have been achieved. Capacity is, of course, dependent on many factors. These include type of material, amount of oversize, amount of undersize, moisture content, particle shape, screen opening size, and screen medium. In general, from 0.40 to 0.75 ft^2 of screen surface area will be needed for every cubic yard of feed handled per hour.

SAMPLE MILL DESIGN

It is not possible to provide complete instruction on mill design within the constraints of this manual. Mills must be planned with the intention of treating the size, shape, and grade characteristics of a specific feed. Sample gold mill flowsheets shown in figures 1, 2, and 3 are included to aid the evaluator in cost estimation only. They are provided to demonstrate that, in most instances, material will have to be fed, washed, sized, and separated for proper recovery.

Tables 1, 2, and 3 provide sample material balances for these mills.

Table 1.—Sample material balance, sluice mill

(Specific gravity: Gold, 17.50; waste, 2.81)

	Feed	Concentrate	Tails
Rate .... yd^3/d ..	120	0.1	119.9
Composition .... wt % ..	100	0.08	99.92
Specific gravity	2.81	2.82	2.81
Grade .... tr oz Au/yd^3 ..	0.040	42.24	0.005
Gold distribution:			
tr oz/d	4.8	4.224	0.576
%	100	88	12

Table 2.—Sample material balance, jig mill

(Specific gravity: Gold, 17.50; waste, 2.65)

	Feed	Concentrate	Tails
Rate .... yd^3/d ..	700	0.1	699.9
Composition .... wt % ..	100	0.01	99.99
Specific gravity	2.65	2.71	2.65
Grade .... tr oz Au/yd^3 ..	0.030	199.50	0.002
Gold distribution:			
tr oz/d	21.0	19.95	1.05
%	100	95	5

Table 3.—Sample material balance, table mill

(Specific gravity: Gold, 17.50; waste, 2.73)

	Feed	Concentrate	Tails
Rate .... yd^3/d ..	250	0.2	249.8
Composition .... wt % ..	100	0.08	99.92
Specific gravity	2.73	2.75	2.73
Grade .... tr oz Au/yd^3 ..	0.045	53.44	0.002
Gold distribution:			
tr oz/d	11.25	10.688	0.562
%	100	95	5

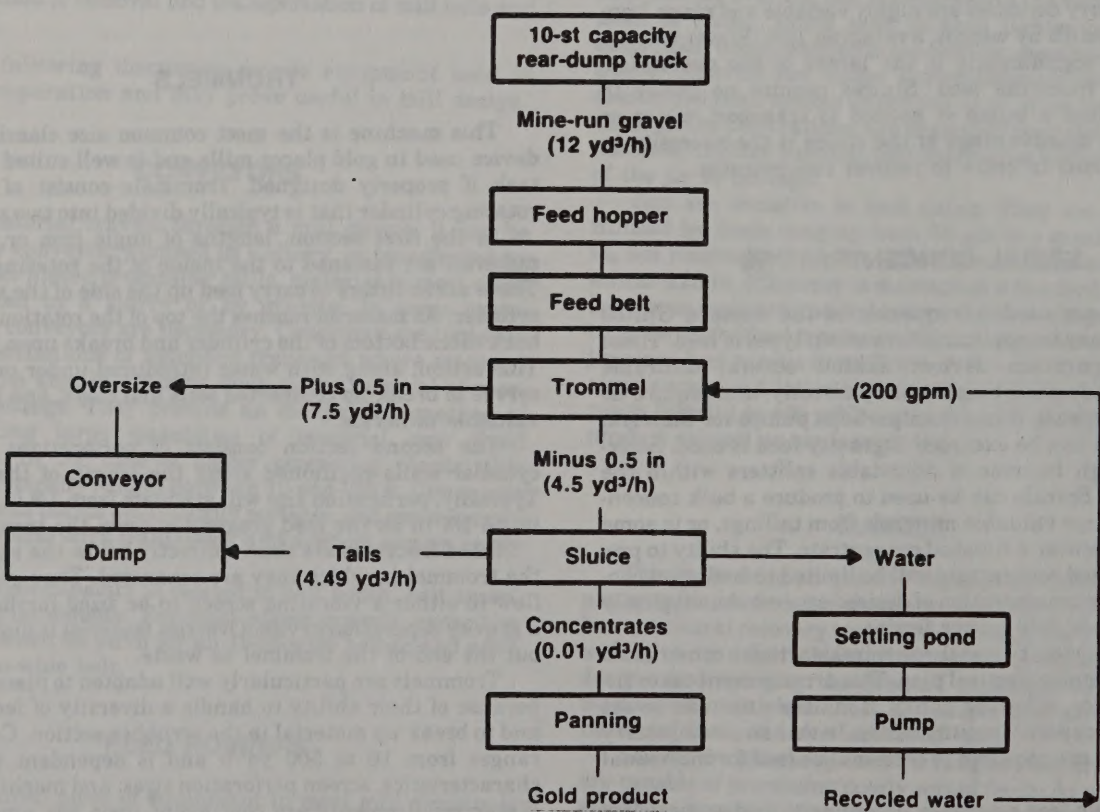

Figure 1.—Sample flow sheet, sluice mill.

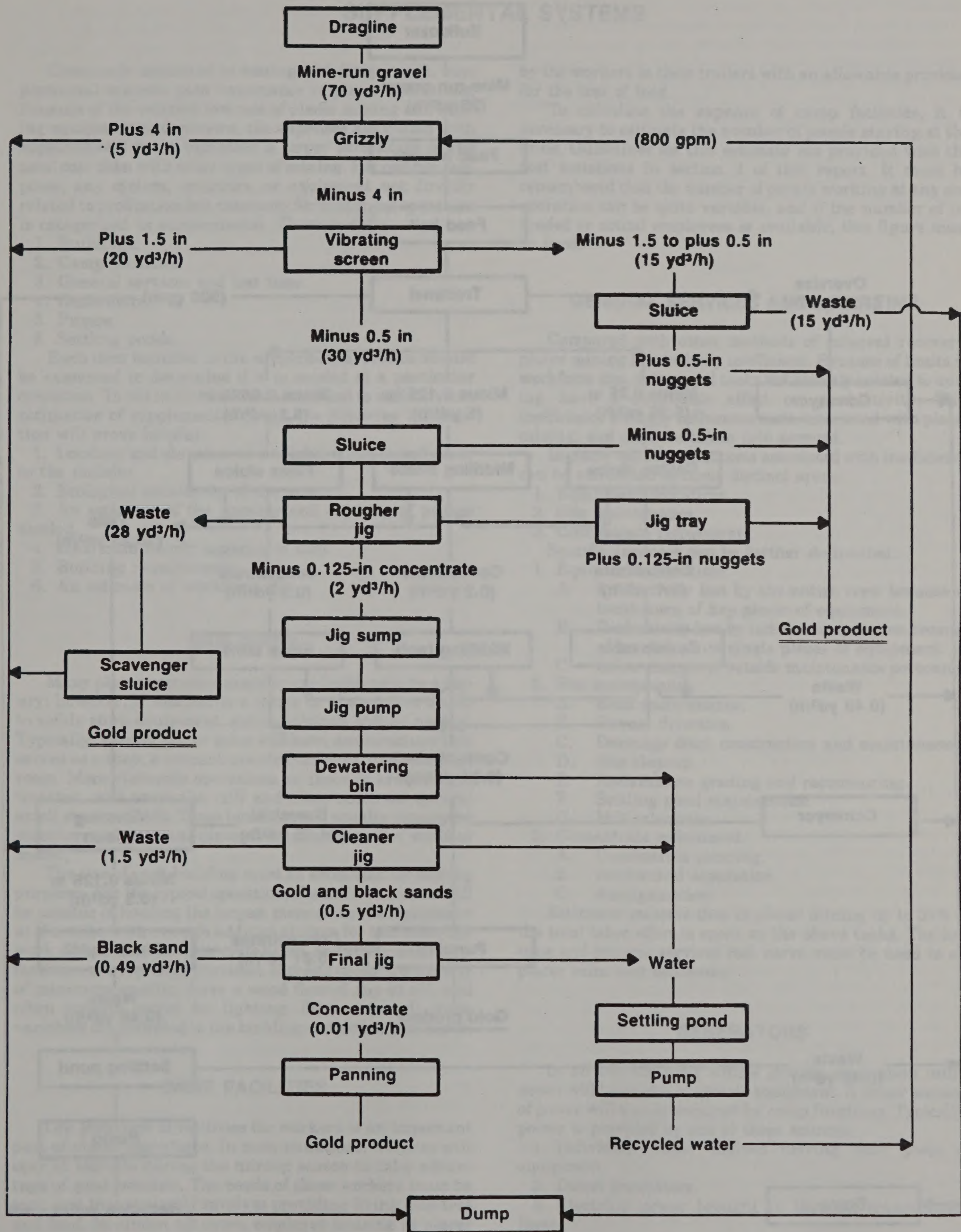

Figure 2.—Sample flow sheet, jig mill.

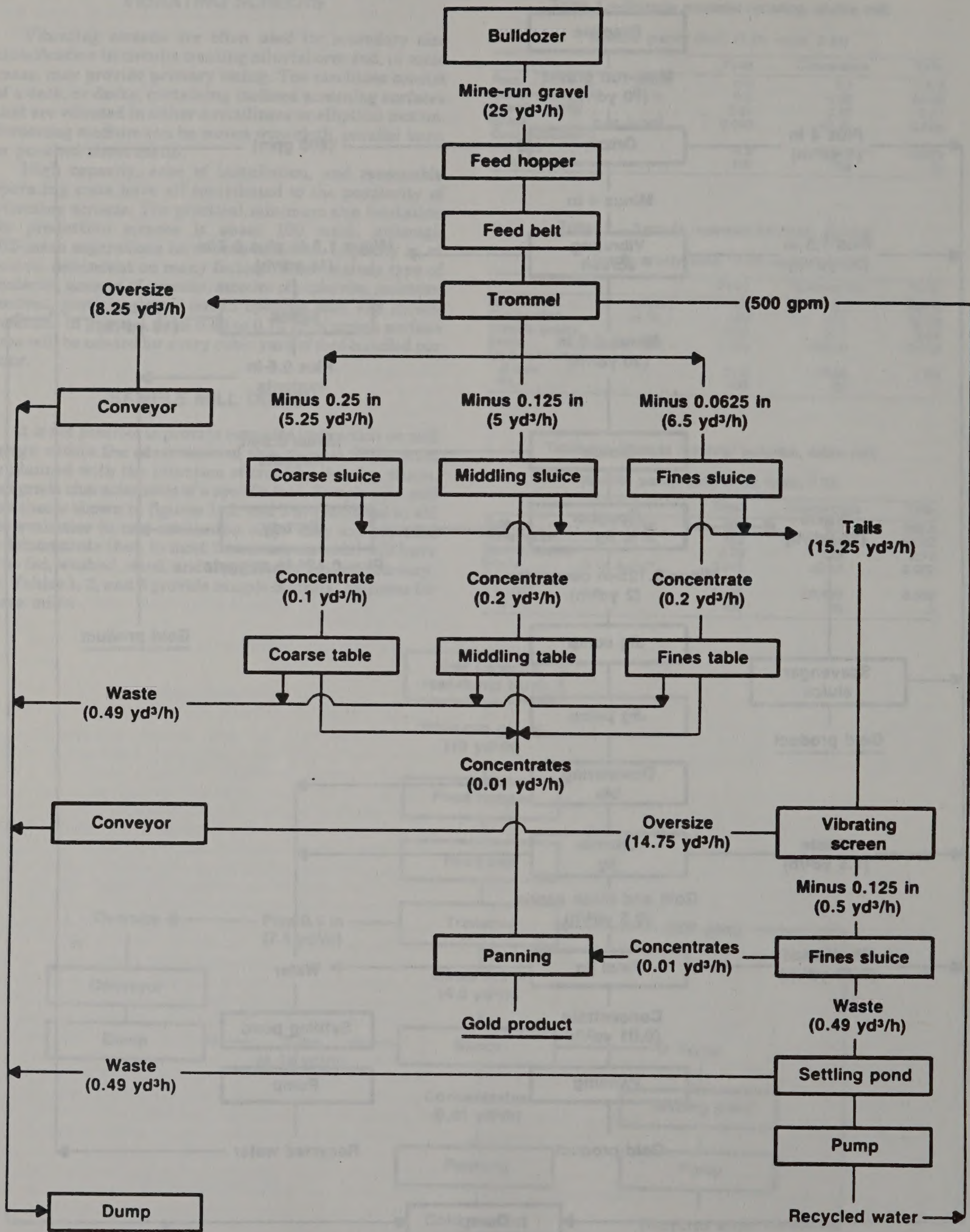

Figure 3.—Sample flow sheet, table mill.

SUPPLEMENTAL SYSTEMS

Commonly neglected in costing and design work, supplemental systems gain importance in placer operations. Because of the relative low cost of placer mining and milling equipment and systems, the expenses associated with supplemental items represent a larger percentage of the total cost than with other types of mining. For costing purposes, any system, structure, or equipment not directly related to production but necessary for continued operation is categorized as supplemental. These include

1. Buildings.
2. Camp facilities.
3. General services and lost time.
4. Generators.
5. Pumps.
6. Settling ponds.

Each item included in the supplemental section should be examined to determine if it is needed at a particular operation. To aid in this determination and to assist in cost estimation of supplemental items, the following information will prove helpful:

1. Location and elevation of available water in reference to the millsite.
2. Ecological sensitivity of the area.
3. An estimate of the number and capacity of pumps needed.
4. Maximum hourly capacity of mill.
5. Building requirements.
6. An estimate of workforce size.

BUILDINGS

Many placer operators consider any building to be a luxury; however, if weather is a factor or if operators desire to safely store equipment, some buildings will be needed. Typically, a small placer mine will have one structure that serves as a shop, a concentrate cleanup area, and a storage room. More elaborate operations, or those in areas of bad weather, will cover the mill and often construct several small storage sheds. These buildings are usually temporary structures of minimal dimensions constructed of wood or metal.

The size of each building must be estimated for costing purposes. For the typical operation, the main structure will be capable of housing the largest piece of mobile equipment at the mine with enough additional room for maintenance work. Shops often have concrete floors, and power and water facilities are typically provided. Storage sheds are usually of minimum quality, have a wood floor if any at all, and often contain power for lighting. Factors for all these variables are provided in the building cost estimation curve.

CAMP FACILITIES

The provision of facilities for workers is an important part of placer operations. In most situations, workers will stay at the site during the mining season to take advantage of good weather. The needs of these workers must be met, and that typically involves providing living quarters and food. In almost all cases, employee housing at placer mines consists of mobile homes or trailers with a minimum amount of support equipment. Cooking is generally done by the workers in their trailers with an allowance provided for the cost of food.

To calculate the expense of camp facilities, it is necessary to estimate the number of people staying at the mine. Guidelines for this estimate are provided with the cost equations in section 2 of this report. It must be remembered that the number of people working at any one operation can be quite variable, and if the number of intended or actual employees is available, this figure must be used.

GENERAL SERVICES AND LOST TIME

Compared with other methods of mineral recovery, placer mining is relatively inefficient. Because of limits in workforce size, delays and tasks not directly related to mining have a noticeable effect on productivity. This inefficiency strongly influences costs associated with placer mining, and must be taken into account.

In placer mining, most costs associated with inefficiency can be attributed to three distinct areas:

1. Equipment downtime.
2. Site maintenance.
3. Concentrate refinement.

Specific expenses can be further delineated.

1. Equipment downtime.
   - A. Productivity lost by the entire crew because of breakdown of key pieces of equipment.
   - B. Productivity lost by individual operators because of breakdown of single pieces of equipment.
   - C. Labor charges of outside maintenance personnel.
2. Site maintenance.
   - A. Road maintenance.
   - B. Stream diversion.
   - C. Drainage ditch construction and maintenance.
   - D. Site cleanup.
   - E. Reclamation grading and recontouring.
   - F. Settling pond maintenance.
   - G. Mill relocation.
3. Concentrate refinement.
   - A. Concentrate panning.
   - B. Mechanical separation.
   - C. Amalgamation.

Estimates indicate that in placer mining up to 37% of the total labor effort is spent on the above tasks. The lost time and general services cost curve must be used in all placer mine cost estimates.

GENERATORS

In all but the most simple gravity separation mills, power will be needed to operate equipment. A minor amount of power will also be required for camp functions. Typically, power is provided by one of three sources:

1. Individual diesel engines driving each piece of equipment.
2. Diesel generators.
3. Electrical power brought in through transmission lines.

The third source generally requires excessive initial capital expenditures. Transmission lines are considered only

when the mill capacity is well over 200 yd^3/h, existing transmission lines are located near the site, or the mine life is expected to be 15 yr or more. Power source selection should be based on lowest overall cost and minimum environmental impact. For most small- to medium-sized gravity separation mills in remote locations, diesel generators are selected to provide power.

Cost estimation curves in this report are based on diesel generators providing all power to mill equipment. Electric power costs contained in individual processing equipment operating cost curves account for diesel generator operating costs.

PUMPS

Water, used to wash gravel and to initiate slurrying of the feed, is typically introduced as gravel enters the trommel or screen. More water is added as needed throughout the circuit to dilute the slurry or assist in washing. To provide adequate washing, this water must be introduced under pressure which, in many cases, necessitates the use of pumps. Pumps will also be needed if mill water is to be recycled through settling ponds. Under certain circumstances, one pump can handle all tasks required in a placer processing plant utilizing recycled water. It is preferable to minimize the use of pumps by taking advantage of gravity.

Water use is dependent on several factors, including

1. Washing required to properly slurry feed.
2. Type of separation equipment used.
3. Availability of water.
4. Size and nature of valuable mineral constituents.

For costing purposes, the evaluator must estimate the volume of water pumped per minute and the pumping head. A separate estimate must be made for each pump. Water requirements can either be calculated using parameters given in the processing portion of section 1, or roughly estimated using the following equation:

$$\text{Water consumption (gpm)} = 94.089(X)^{0.546},$$

where X = maximum cubic yards of mill feed handled per hour.

This equation provides the total gallons of water per minute consumed by the mill. Although not technically accurate, for the purposes of this report, head may be estimated as the elevation difference between the pipe outlet at the mill or upper settling pond, and the pump intake.

SETTLING PONDS

With the current level of environmental awareness, it is almost assured that mill water will have to be treated prior to discharge. Placer mines typically recycle mill effluent through one or more settling ponds to control environmental impact.

To calculate the cost of settling pond construction using this report, only the maximum mill feed rate is required. Cost curves provide the construction expense of unlined ponds sized to comply with most regulations. In some instances, the pond will have to be lined with an impervious material. This is often required in ecologically sensitive areas, or in situations where underlying soils do not properly filter mill effluent, thereby increasing the turbidity of nearby streams. A factor is provided in the settling pond cost curve for impervious linings.

ENVIRONMENT

Enviromental costs are often decisive in placer mine economic feasibility. Costs associated with water quality control and aesthetics are inescapable and can represent a significant percentage of total mining expenses. Methods to minimize ecological disturbance are now considered an integral function of the mining sequence and are treated as such in cost estimation.

Stream siltation from mill effluent and land disturbance from excavation are the main environmental problems facing placer miners. Reduction of water quality is often the biggest problem, and many techniques have been devised to lessen the impact caused by mill operation. One methcd involves limiting mill operation to short periods of time, thus allowing effluent to disperse before additional mill discharge is introduced. Often the mill is designed with the intent of using as little water as possible for valuable mineral separation. The most common solution is mill water recirculation facilitated by the construction of settling ponds. These ponds are used to hold mill effluent until particulate matter has settled; water from the ponds is then reused in the mill circuit.

Mining of alluvial deposits necessitates disturbance of large areas of land. Typically all trees, brush, grasses, and ground cover will be cleared. This task alone may present a major stumbling block, because some States restrict open burning. Next, a layer of overburden is removed to expose the deposit. Finally, the valuable mineral-bearing gravel can be excavated.

Current technology suggests that control of land disturbance be incorporated into the mining sequence. Mill tailings and oversize are typically dumped back into worked-out areas. Soil cover and overburden are removed just prior to pay gravel excavation, then hauled to mined-out areas to be graded and contoured over replaced tails. Often the surface is revegetated. In most instances, the operator will have no choice but to implement ecological control and reclamation procedures. Operators are typically required to post a bond to cover the cost of reclaiming mined lands, and if the surface is left disturbed, these bonds will be forfeited.

Regulations vary from State to State, and may appear difficult and confusing at first; however, by contacting information services at State capitals, operators will be directed to the agencies concerned. These agencies will detail regulations concerning placer operation and will also point out which Federal agencies might be involved (U.S. Forest Service or U.S. Bureau of Land Management). In most instances, contact will have to be made with both State

and Federal agencies. Typically, meeting environmental requirements for the State will satisfy Federal regulations.

As stated earlier, environmental control is an integral part of mine and mill design, and costs are treated accordingly. Equations are provided for calculating the cost of mill tails and oversize placement. Expenses associated with grading and contouring are contained in the lost time and general services curve. An equation is also provided for the construction of settling ponds, if water is to be recycled.

Bond costs are not included since requirements are highly variable. One other cost may arise that is not covered in section 2. This is the expense of replanting, and usually ranges from $100 to $200 per acre.

COST ESTIMATION

After selecting exploration, mining, milling, and supplemental techniques, the next step in cost estimation is the choice of appropriate cost curves. If the evaluator has completed the mine design prior to attempting cost estimation, this task consists of simply going through section 2 of this report and selecting the proper equations. The list of capital and operating categories at the beginning of section 2 will aid in choosing individual curves.

Costs used in deriving the estimation equations were collected from several sources. These include

1. Placer mine operators.
2. Mine equipment suppliers.
3. Published cost information services.

In all cases, cost figures quoted in the text and points used in cost equation derivations are averages of all data available. A bibliography of cost information publications follows section 2. Many of these sources contain both cost and capacity information and can be used to supplement this manual.

Cost estimation methodology in this handbook is based on the Bureau's Cost Estimation System (CES), first published in 1977 as "Capital and Operating Cost Estimation System Handbook," by STRAAM Engineers, Inc. Procedures for cost estimation using this report closely follow that publication. The cost estimation portion of this report is divided into operating and capital costs. Cost equations are similar for both with the only difference appearing in the units of the final answer. Capital costs are given in total dollars expended and operating costs in dollars per year.

Using the appropriate curves, a separate cost is calculated for each capital and operating cost item. Only costs directly associated with the operation under evaluation need be calculated. All other cost items should be ignored. After calculation, item costs should be entered on the respective capital and operating cost summary forms (see figures 5 and 6 in section 2).

Upon summation of individual expenses, a contingency may be added to both capital and operating costs. It is difficult to anticipate every condition that may arise at a particular operation, and the purpose of the contingency is to account for unforeseen expenditures. This figure is typically based on the degree of certainty of the evaluation in relation to available information, and ranges from 10% to 20%.

Cost per cubic yard of pay gravel processed is determined by dividing the sum of all annual operating costs by the total amount of pay gravel processed per year. Summation of individual capital expenditures produces the total capital cost.

Use of the individual curves is described in the following paragraphs.

COST EQUATIONS

Capital and operating costs are divided into labor, equipment, and supply categories. One, two, or all three of these categories will be present in each cost equation. The sum of costs from each of these categories provides the total cost for any single cost item. To facilitate cost adjustments respective to specific dates, the labor, equipment, and supply classifications are further broken down into subcategories.

Typically, each cost item will have a number of site adjustment factors. These are provided to account for characteristics specific to a particular deposit. These factors determine the precision of the final cost, so they must be selected and used carefully. Assistance in determining the correct use of a factor, or in understanding the parameters involved in a cost item, may be found in the preceding pages.

To further improve cost estimates, labor rates are also adjustable. Rates can vary greatly for small placer operations. For this reason, adjustments can be made to the fixed rates used in this report for specific known rates at individual operations.

COST DATE ADJUSTMENTS

All cost equations were calculated in January 1985 dollars. Costs calculated for any particular cost item are broken down into specific categories and subcategories to facilitate adjustment to specific dates. These include

Labor.
1. Mine labor.
2. Processing labor.
3. Repair labor.

Equipment.
1. Equipment and equipment parts.
2. Fuel and lubrication.
3. Electricity.
4. Tires.

Supplies.
1. Steel items.
2. Explosives.
3. Timber.
4. Construction materials.
5. Industrial materials.

For placer mining, most general maintenance and nonoverhaul repairs are accomplished by the equipment operator, so repair labor rates are assumed to be equal to those of the operator. If information available to the evaluator indicates that this is not the case, repair labor

portions of the total labor cost are stated to facilitate adjustment.

Equipment operating costs are broken down into respective percentages contributed by parts, fuel and lubrication, electricity, and tires. These percentages, listed immediately following the cost equations, are used to calculate specific costs for each subcategory so that they may be updated. Supply costs are broken down and handled in a similar manner.

Cost date indexes for the preceding subcategories are provided in table 4. These and other cost indexes are updated every 6 months and are available from the Bureau of Mines, Western Field Operations Center, East 360 Third Avenue, Spokane, WA 99202. To adjust a cost to a specific date, divide the index for that date by the index for January 1985, and multiply the resulting quotient by the cost calculated for the respective subcategory. An example of such an update follows.

Example Cost Update

Calculate the cost in July 1985 dollars of extracting and moving pay gravel 300 ft over level terrain using bulldozers. Assume a 200-LCY/h operation, and use the operating cost equations provided in the operating costs—mining-bulldozers portion of section 2.

Operating costs per LCY (from section 2):

Equipment operating cost	$0.993(200)^{-0.430}$ =	$0.102
Labor operating cost	$14.01(200)^{-0.945}$ =	.094
January 1985 total		.196

Subcategory costs per LCY (from section 2):

Equipment parts	0.47 × $0.102 =	$0.048
Fuel and lubrication	0.53 × $0.102 =	$0.054
Operator labor	0.86 × $0.094 =	$0.081
Repair labor	0.14 × $0.094 =	$0.013

Update indexes (from table 4):	Subcategory	July 85/Jan. 85	Quotient
Equipment parts	Equipment ..	362.3/360.4	1.005
Fuel and lubrication ..	Fuel	630.7/636.2	0.991
Operator labor	Mine labor ..	$11.98/$11.69	1.025
Repair labor	Mine labor ..	$11.98/$11.69	1.025

Updated costs per LCY:

Equipment parts	1.005 × $0.048 =	$0.048
Fuel and lubrication	0.991 × $0.054 =	.054
Operator labor	1.025 × $0.081 =	.083
Repair labor	1.025 × $0.013 =	.013
July 1985 total cost per LCY		.198

SITE ADJUSTMENT FACTORS

As stated earlier, adjustment factors determine the precision for cost estimates and must be used carefully. Several factors are provided for each curve, and their use will significantly alter the calculated cost. The following example illustrates factor use.

Example Adjustment Factor Application

Calculate the cost of extracting pay gravel in a hard digging situation and moving it 800 ft up an 8% gradient using bulldozers. Assume a 200-LCY/h operation (January 1985 dollars), and use the operating cost and adjustment factor equations provided in the operating costs—mining-bulldozers portion of section 2.

Operating costs per LCY (from section 2):

Equipment operating cost	$0.993(200)^{-0.430}$ =	$0.102
Labor operating cost	$14.01(200)^{-0.945}$ =	.094
January 1985 total		.196

Factors (from section 2):

Distance	$F_D = 0.00581(800)^{0.904}$ =	2.447
Gradient	$F_G = 1.041e^{[0.015(8.0)]}$ =	1.174
Digging difficulty		1.670
Used equipment:		
Equipment	$U_e = 1.206(200)^{-0.013}$ =	1.126
Labor	$U_l = 0.967(200)^{0.015}$ =	1.047

Factored cost per LCY:
From total cost equation for bulldozers:

[$0.102(1.126) + $0.094(1.047)] × 2.447 × 1.174 × 1.670 =

January 1985 total cost per LCY $1.023

The 500% increase in operating cost, from $0.196 to $1.023 per loose cubic yard, demonstrates the dramatic effect of using the proper factors. If a cost category contains a factor not applicable to the deposit in question, then simply leave that factor out of the total cost equation.

The variables inserted in the factor equations are generally self-evident. An exception to this is the digging difficulty factor. Parameters for this factor are based on the following:

1. Easy digging.—Unpacked earth, sand, and gravel.
2. Medium digging.—Packed earth, sand, and gravel, dry clay, and soil with less than 25% rock content.
3. Medium to hard digging.—Hard packed soil, soil with up to 50% rock content, and gravel with cobbles.
4. Hard digging.—Soil with up to 75% rock content, gravel with boulders, and cemented gravels.

It can be seen from these parameters that many deposits will fall into one of the last two categories. Digging difficulty has a dramatic effect on the cost of extraction, so these factors must be chosen carefully.

Bulldozer and backhoe curves both contain a digging difficulty factor. Other excavation equipment, such as draglines, scrapers, and front-end loaders, are generally suited for special digging conditions and are not used in harder ground. Consequently, no digging difficulty factor is provided for these.

Table 4.—Cost date indexes[1]

	Mining labor[2]	Equipment and repair parts	Fuel and lubrication	Electricity	Tires	Bits and steel	Explosives	Timber	Construction material[3]	Industrial material
1960 ...	$2.61	85.8	95.5	100.1	113.1	97.1	94.5	92.1	99	95.3
1961 ...	2.64	87.3	97.2	100.7	109.9	97.2	97.0	87.4	97	94.8
1962 ...	2.70	87.5	96.1	101.9	94.7	95.8	97.0	89.0	96	94.8
1963 ...	2.75	89.0	95.1	101.1	96.9	95.7	100.4	91.2	98	94.7
1964 ...	2.81	91.2	90.7	100.3	97.6	97.0	100.0	92.9	98	95.2
1965 ...	2.92	93.6	92.8	100.3	98.8	97.9	99.6	94.0	97	96.4
1966 ...	3.05	96.5	97.4	99.8	101.3	98.7	98.1	100.1	99	98.5
1967 ...	3.19	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100	100.0
1968 ...	3.35	105.7	98.1	100.9	102.7	101.9	102.3	117.4	127	102.5
1969 ...	3.61	110.4	99.6	102.2	98.3	107.0	104.7	131.6	108	106.0
1970 ...	3.85	115.9	101.0	106.6	105.4	115.1	106.7	113.7	109	110.0
1971 ...	4.06	121.4	106.8	115.5	110.3	121.8	113.3	135.5	133	114.0
1972 ...	4.41	125.7	108.9	123.9	111.3	128.4	115.2	159.4	151	117.9
1973 ...	4.73	130.7	128.6	132.6	115.7	136.2	120.1	205.2	154	125.9
1974 ...	5.21	152.3	223.4	172.3	141.6	178.6	150.0	207.1	167	153.8
1975 ...	5.90	185.2	257.5	209.7	155.4	200.9	178.0	192.5	186.3	171.5
1976 ...	6.42	198.9	276.6	226.7	172.8	215.9	187.2	233.0	205.5	182.4
1977 ...	6.88	213.7	308.1	257.0	181.5	230.3	193.1	276.5	237.7	195.1
1978 ...	7.67	232.8	321.0	279.5	192.0	253.5	208.7	322.1	247.7	209.4
1979 ...	8.50	256.2	444.8	305.3	219.6	283.5	225.6	354.3	269.28	236.5
1980[4] ..	8.70	275.4	582.4	334.8	236.9	297.3	237.1	336.3	280.86	260.3
1980[5] ..	9.08	290.9	693.3	376.0	250.4	300.4	254.4	327.3	289.05	275.6
1981[4] ..	9.78	304.9	736.0	393.9	256.2	322.8	268.5	331.6	298.25	289.9
1981[5] ..	10.07	324.0	818.4	429.9	269.6	338.7	292.8	330.1	312.11	306.0
1982[4] ..	10.58	337.0	802.9	454.0	271.6	343.1	293.2	310.6	324.74	311.7
1982[5] ..	10.91	346.1	777.1	471.5	272.6	337.4	294.8	319.2	330.56	313.0
1983[4] ..	11.10	348.6	727.1	482.6	285.4	333.2	300.4	324.2	342.01	314.0
1983[5] ..	11.31	352.7	694.9	492.2	256.6	341.3	302.8	372.5	357.28	316.6
1984[4] ..	11.56	354.3	669.7	492.0	258.0	354.1	301.3	353.2	355.52	319.2
1984[5] ..	11.62	358.2	674.6	525.5	256.3	357.2	312.4	343.3	357.90	324.0
1985[6] ..	11.69	360.4	636.2	524.9	262.0	357.4	313.4	343.2	358.32	323.2
1985[5] ..	11.98	362.3	630.7	540.3	246.0	354.6	312.1	354.9	363.63	324.3

[1] Unless otherwise noted, based on U.S. Bureau of Labor Statistics (BLS) "Producer Price Indexes," base year 1967 = 100.
[2] Based on BLS "Employment and Earnings: Average Hourly Earnings, Mining."
[3] Based on Engineering and News Record "Market Trends: Building Cost."
[4] January.
[5] July.
[6] January (base cost year for this report).

LABOR RATES

The cost of labor in placer mining is highly variable and cannot be precisely estimated in every case. For the purposes of this report, only two separate labor rates are used: \$15.69/h for mining functions, and \$15.60/h for milling. These rates apply to operation, maintenance, installation, and construction labor. The labor portions of each specific cost category are broken out and in this way can be adjusted to the estimator's particular labor rate. To accomplish this, multiply the labor cost for each category by the ratio of desired labor rate to mining or milling labor rate (\$15.69/h or \$15.60/h). The following example illustrates this adjustment.

Example Labor Rate Adjustment

Calculate the cost of extracting and moving pay gravel 300 ft over level terrain using bulldozers with an operator labor cost of \$18.00/h. Assume a 200-LCY/h operation (January 1985 dollars), and use the operating cost equations provided in the operating costs—mining-bulldozers portion of section 2.

Operating costs per LCY (from section 2):

Equipment operating cost	$0.993(200)^{-0.430}$ =	\$0.102
Labor operating cost	$14.01(200)^{-0.945}$ =	.094
January 1985 total		.196

Labor adjustment:

Labor operating cost per LCY .... (\$18.00/\$15.69) × \$0.094 =	.108

Adjusted cost per LCY:

Equipment operating cost	.102
Labor operating cost	.108
January 1985 total cost per LCY	.210

Labor rates are based on wage scales for the western United States (including Alaska) and include a 24% burden. This burden consists of 9.8% workers compensation insurance, 7.0% Social Security tax, 3.7% State unemployment insurance, and 3.5% Federal unemployment tax. If other costs such as health and retirement benefits are to be included, they must be added to an estimated labor rate.

To familiarize the reader with the use of this cost estimating system, an example of a complete cost estimate is included in the appendix.

FINANCIAL ANALYSIS

The purpose of this report is to provide an estimate of capital and operating costs for small placer mines. A distinction must be made between a cost estimate and an economic feasibility analysis. Capital and operating costs are simply two separate variables in a complete economic analysis. To determine the economic feasibility of an operation, the evaluator must consider each of the following:

1. Recoverable value of commodity.
2. Local, State, and Federal taxes.
3. Capital depreciation.
4. Depletion allowances.
5. Desired return on investment.
6. Costs and methods of project financing.
7. Inflation.
8. Escalation.
9. Environmental intangibles.

Economic feasibility analysis is a subject in itself, and will not be covered here. The preceding list is included to emphasize the following: *A prospect is not economically feasible simply because the apparent commodity value exceeds the total capital and accrued operating costs calculated from this manual.*

The costs associated with the preceding list are real and must be considered when determining the feasibility of a prospect. Any attempt to provide guidelines for determination of feasibility based solely on estimates of capital and operating costs would be highly misleading. There is no quick and easy way to account for the wide variety of situations encountered in economic analysis. Each one of the preceding items must be examined individually to provide accurate economic feasibility estimates, and a complete cash-flow analysis is the only way to ensure that proper results are obtained. To accomplish this, *all* yearly income and expenses must be tabulated. Then the rate of return over time must be calculated from the resultant profits or losses. The evaluator must consider all factors influencing income and include all expenses as well as account for the value of money over time and choose an acceptable rate of return.

In brief, the operator will have to receive adequate revenues from commodities recovered to

1. Cover all operating expenses.
2. Recover initial equipment expenditures.
3. Provide for equipment replacement.
4. Cover all exploration and development costs.
5. Pay taxes.
6. Compensate for inflation and cost escalation.
7. Supply a reasonable profit.

Only when enough revenue is produced to cover all of the above can an operation be considered economically feasible.

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SECTION 2.—COST ESTIMATION

CAPITAL AND OPERATING COST CATEGORIES

Section 2 contains equations for estimating capital and operating costs associated with placer mining. Equations are provided for the following items.

Capital costs:
- Exploration:
  - Panning
  - Churn drilling
  - Bucket drilling
  - Trenching
  - General reconnaissance
  - Camp costs
  - Seismic surveying
  - Rotary drilling
  - Helicopter rental
- Development:
  - Access roads
  - Clearing
- Preproduction overburden removal:
  - Bulldozers
  - Draglines
  - Front-end loaders
  - Rear-dump trucks
  - Scrapers
- Mine equipment:
  - Backhoes
  - Bulldozers
  - Draglines
  - Front-end loaders
  - Rear-dump trucks
  - Scrapers
- Processing equipment:
  - Conveyors
  - Feed hoppers
  - Jig concentrators
  - Sluices
  - Spiral concentrators
  - Table concentrators
  - Trommels
  - Vibrating screens
- Supplemental:
  - Buildings
  - Employee housing
  - Generators
  - Pumps
  - Settling ponds

Operating costs:
- Overburden removal:
  - Bulldozers
  - Draglines
  - Front-end loaders
  - Rear-dump trucks
  - Scrapers
- Mining:
  - Backhoes
  - Bulldozers
  - Draglines
  - Front-end loaders
  - Rear-dump trucks
  - Scrapers
- Processing:
  - Conveyors
  - Feed hoppers
  - Jig concentrators
  - Sluices
  - Spiral concentrators
  - Table concentrators
- Tailings removal:
  - Bulldozers
  - Draglines
  - Front-end loaders
  - Rear-dump trucks
  - Scrapers
- Trommels
- Vibrating screens
- Supplemental:
  - Employee housing
  - Generators
  - Lost time and general services
  - Pumps

Included in this section are summary forms (figs. 4-6) that may be used to aid in total capital and operating cost calculations. A bibliography of cost information sources is provided at the end of this section.

The appendix contains a complete sample cost estimation. This sample will familiarize the reader with cost estimation techniques used in this report.

EXPLORATION

Two methods are presented for calculating exploration costs. Method 1 allows the evaluator to roughly estimate costs with a minimum of information. Method 2 requires a detailed exploration plan and provides the user with a much more precise cost.

Method 1: If information concerning exploration of a deposit is not available, the following equation may be used to estimate an exploration capital cost. It must be emphasized, however, that *costs calculated from this equation can be very misleading,* and it is recommended that a detailed exploration program be designed if possible and that costs be assigned using method 2.

As stated in section 1, the amount of exploration required is a highly variable function of many factors. This equation is based on estimated exploration costs for several successful placer operations, but these deposits may have little in common with the one being evaluated.

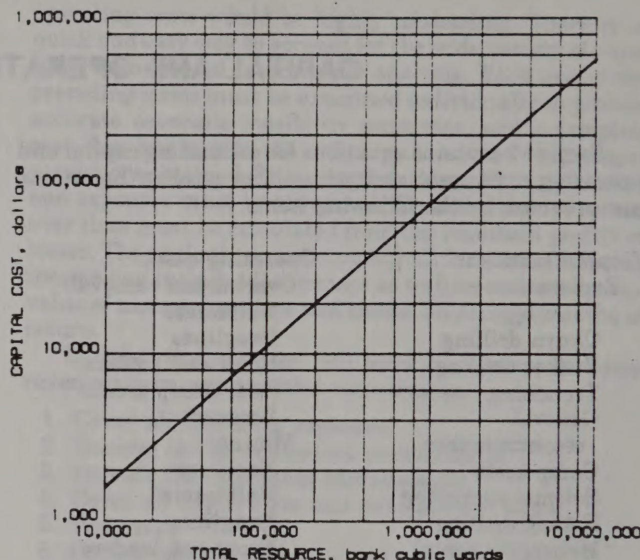
The base equation is applied to the following variable:

X = Total estimated resource, in bank cubic yards (BCY)

Base Equation:

Exploration capital costs . . $Y_C = 0.669(X)^{0.849}$

An exact breakdown of expenses included in this cost is not available. In general, exploration is a labor-intensive task. Unless the deposit is extremely remote, a large share of the exploration cost will be attributed to labor. If the deposit is remote, costs of access equipment (helicopters, etc.) will become a factor.



Exploration capital costs

Method 2: Excellent cost data for most exploration functions may be found in the Bureau's Cost Estimation System (CES) Handbook (IC 9142). Functions covered in that publication include

Helicopter rental rates.
Sample preparation and analysis costs.
Drill capacities and costs for core, rotary, and hammer drills.
Survey charges.
Labor rates.
Travel costs.
Ground transportation costs.
Field equipment costs.
Geological, geophysical, and geochemical exploration technique costs.

Costs directly related to placer mining from the above list are summarized in the following tabulations. Several items particular to placer mining are not covered in the CES Handbook. These items, for which costs follow, include

Panning.
Churn drilling.
Bucket drilling.
Trenching.

Exploration Cost Tabulations: As in the CES Handbook, costs are given in dollars per unit processed (cubic yard, sample, foot drilled, etc.). The product of the unit cost and the total units processed constitutes the total capital cost for any particular method of exploration. Total exploration costs consist of the sum of these individual exploration method expenses. A summary sheet for these calculations is shown in figure 4.

EXPLORATION—PANNING

Average cost per sample	$2.10
Cost range	$1.90–$2.60
Cost variables	Labor efficiency and material being panned.

EXPLORATION—CHURN DRILLING

Average cost per foot	$45
Cost range	$20–$70
Cost variables	Depth of hole, material being drilled, site access, and local competition.

EXPLORATION—BUCKET DRILLING

Average cost per foot	$9.20
Cost range	$5–$20
Cost variables	Depth of hole, material being drilled, and site access.

EXPLORATION—TRENCHING

Average cost per cubic yard	$7.10
Cost range	$2.25–$28.50
Cost variables	Labor efficiency, material being sampled, site access, equipment ownership, sampling method, and total volume of work to be done.

CES Exploration Cost Tabulations: Some of the more pertinent exploration cost items presented in the CES Handbook (IC 9142) are summarized in the following. A detailed description of these items can be found in that publication.

EXPLORATION—GENERAL RECONNAISSANCE

Average cost per worker-day	$195
Cost range	$175–$210
Cost variables	Deposit access, terrain, and labor efficiency.

EXPLORATION—CAMP COSTS

Average cost per worker-day	$30
Cost range	$19–$41
Cost variables	Deposit remoteness, terrain, access, and climate.

EXPLORATION—SEISMIC SURVEYING (REFRACTION)

Average cost per linear foot	$1.50
Cost range	$1.00–$2.50
Cost variables	Labor efficiency, deposit access, and terrain.

EXPLORATION—ROTARY DRILLING

Average cost per foot	$6.50
Cost range	$2.00–$11.50
Cost variables	Depth of hole, material being drilled, and site access.

EXPLORATION—HELICOPTER RENTAL

Average cost per hour	$395
Cost range	$305–$590
Cost variables	Passenger capacity, payload capacity, cruise speed, and range.

EXPLORATION COST SUMMARY FORM

Capital cost calculation:

General reconnaissance	______	worker-days	× $ ______	/worker-day	=	______
Camp costs	______	worker-days	× $ ______	/worker-day	=	______
Panning	______	samples	× $ ______	/pan	=	______
Churn drilling	______	ft drilled	× $ ______	/ft	=	______
Bucket drilling	______	ft drilled	× $ ______	/ft	=	______
Trenching	______	yd³	× $ ______	/yd³	=	______
Seismic surveying	______	linear ft	× $ ______	/linear ft	=	______
Rotary drilling	______	ft drilled	× $ ______	/ft	=	______
Helicopter time	______	h	× $ ______	/h	=	______
______	______		× $ ______	/______	=	______
______	______		× $ ______	/______	=	______
______	______		× $ ______	/______	=	______
______	______		× $ ______	/______	=	______
______	______		× $ ______	/______	=	______
______	______		× $ ______	/______	=	______
______	______		× $ ______	/______	=	______
______	______		× $ ______	/______	=	______
Total						______

Figure 4.—Exploration cost summary form.

DEVELOPMENT—ACCESS ROADS

Capital Cost Equation: This equation provides the cost per mile of road construction to the deposit and between various facilities. Costs include clearing and excavation, but do not account for any blasting or gravel surfacing that may be required. The equation is applied to the following variable:

X = Average width of roadbed, in feet.

The following assumptions were made in estimating road costs:

1. Side slope, 25%.
2. Moderate ground cover.
3. Moderate digging difficulty.

Base Equation:

Access road capital cost .... $Y_C = 765.65(X)^{0.922}$

The capital cost consists of 68% construction labor, 13% parts, 16% fuel and lubricants, and 3% tire replacement.

Brush Factor: The original equation is based on the assumption that ground cover consists of a mixture of brush and trees. If vegetation is light (i.e., consisting mainly of brush or grasses), the total cost per mile (covered with brush) must be multiplied by the factor obtained from the following equation:

$F_B = 0.158(X)^{0.325}$.

Forest Factor: If ground cover is heavy (i.e., consisting mainly of trees), the total cost per mile (covered with trees) must be multiplied by the factor obtained from the following equation:

$F_F = 2.000(X)^{-0.079}$.

Side Slope Factor: If average side slope of the terrain is other than 25%, the factor obtained from the following equation must be applied to the total cost per mile:

$F_S = 0.633e^{[0.021 \text{ (percent slope)}]}$.

Surfacing Factor: If gravel surfacing is required, the cost per mile must be multiplied by the following factor to account for the additional labor, equipment, and supply costs:

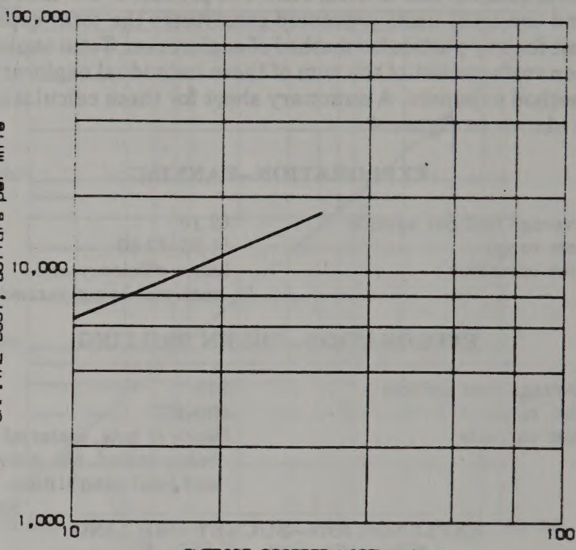
$F_G = 6.743$.

Blasting Factor: In hard-rock situations, blasting may be required. Should this be the case, the cost obtained from the following equation must be *added* to total access road cost.

$F_H = [12{,}059.18(X)^{0.534}] \times$ [miles of roadbed requiring blasting].

Total Cost: Access road capital cost is determined by

$[(Y_C \times F_B \times F_F \times F_S \times F_G) \times \text{total miles}] + F_H$.



Development capital costs - Access roads

This total cost is then entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.

DEVELOPMENT—CLEARING

Capital Cost Equation: This equation provides the total capital cost of clearing brush and timber from the surface of a deposit prior to mining. Costs include labor, equipment, and supplies required to completely strip the surface of growth, and to dispose of debris. The equation is applied to the following variable:

X = Total acreage to be cleared.

The following assumptions were made in estimating clearing costs:

1. Level slope.
2. Moderate ground cover.

Base Equation:

Clearing capital cost .... $Y_C = 1{,}043.61(X)^{0.913}$

The capital cost consists of 68% construction labor, 18% fuel and lubricants, 12% parts, and 2% steel supplies.

Slope Factor: The original equation is based on the assumption that the slope of the surface overlying the deposit is nearly level. If some slope is present, the factor obtained from the following equation must be applied to the clearing capital cost:

$F_S = 0.942e^{[0.008(\text{percent slope})]}$.

Brush Factor: Ground cover is assumed to consist of a mixture of brush and small trees. If the surface is covered with only brush and grasses, the following factor must be applied to the cost:

$F_B = 0.250$.

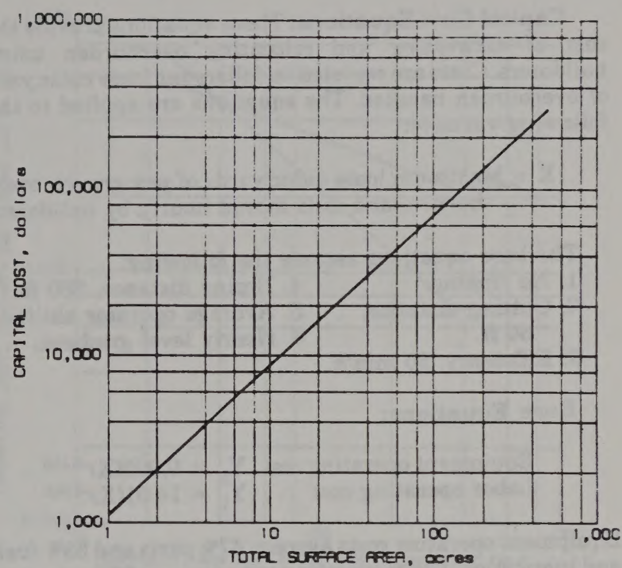
Forest Factor: If the surface is forested, capital cost must be multiplied by the following factor:

$F_F = 1.750$.

Total Cost: Clearing capital cost is determined by

$(Y_C \times F_S \times F_B \times F_F)$.

This total cost is then entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Development capital costs - Clearing

PREPRODUCTION OVERBURDEN REMOVAL—BULLDOZERS

Capital Cost Equations: These equations provide the cost of excavating and relocating overburden using bulldozers. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by bulldozer.

The base equations assume the following:

1. No ripping.
2. Cutting distance, 50 ft.
3. Efficiency, 50 min/h.
4. Dozing distance, 300 ft.
5. Average operator ability.
6. Nearly level gradient.

Base Equations:

Equipment operating cost $Y_E = 0.993(X)^{-0.430}$
Labor operating cost . . . $Y_L = 14.01(X)^{-0.945}$

Equipment operating costs average 47% parts and 53% fuel and lubrication. Labor operating costs average 86% operator labor and 14% repair labor.

Distance Factor: If the average dozing distance is other than 300 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_D = 0.00581(\text{distance})^{0.904}$.

Gradient Factor: If the average gradient is other than level, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_G = 1.041e^{[0.015(\text{percent gradient})]}$

Ripping Factor: If ripping is required, total operating cost must be multiplied by the following factor, this will account for reduced productivity associated with ripping:

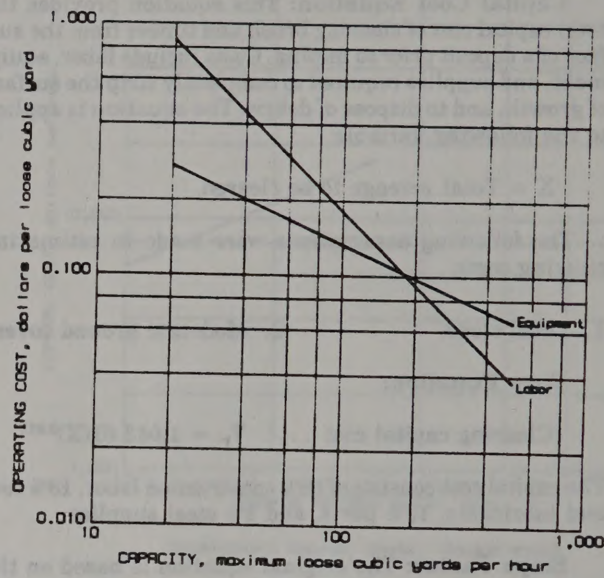
$F_R = 1.595$.

Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor . . . . $U_e = 1.206(X)^{-0.013}$
Labor factor . . . . $U_l = 0.967(X)^{0.015}$

Digging Difficulty Factor: Parameters given in the discussion on site adjustment factors in section 1 should be used to determine if a digging difficulty factor is required. If so, one of the following should be applied to total cost per loose cubic yard:

F_H, easy digging . . 0.830
F_H, medium digging 1.000
F_H, medium-hard digging 1.250
F_H, hard digging . . . 1.670



Overburden removal capital costs - Bulldozers

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_D \times F_G \times F_H \times F_R.$$

To obtain overburden removal capital cost, the total cost per loose cubic yard must be multiplied by total amount of *overburden* handled by bulldozer prior to production. This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.

PREPRODUCTION OVERBURDEN REMOVAL—DRAGLINES

Capital Cost Equations: These equations provide the cost of excavating overburden using draglines. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by dragline.

The base curves assume the following:

1. Bucket efficiency, 0.90.
2. Full hoist.
3. Swing angle, 90°.
4. Average operator ability.

Base Equations:

Equipment operating costs.. $Y_E = 1.984(X)^{-0.390}$
Labor operating costs ..... $Y_L = 12.19(X)^{-0.888}$

Equipment operating costs consist of 67% parts and 33% fuel and lubrication. Labor operating costs consist of 78% operator labor and 22% repair labor.

Swing Angle Factor: If the average swing angle is other than 90°, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_S = 0.304\ (\text{swing angle})^{0.269}$.

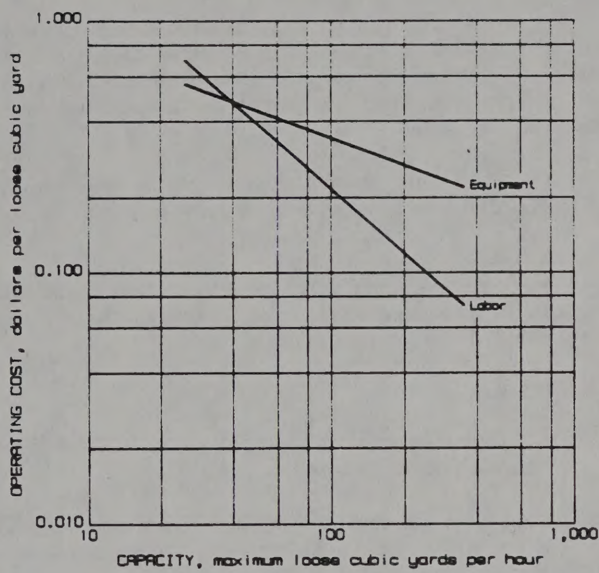
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor .... $U_e = 1.162(X)^{-0.017}$
Labor factor .... $U_l = 0.989(X)^{0.006}$

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_S.$$

To obtain the overburden removal capital cost, the total cost per loose cubic yard must be multiplied by the total amount of *overburden* handled by dragline prior to production. This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Overburden removal capital costs - Draglines

PREPRODUCTION OVERBURDEN REMOVAL—FRONT-END LOADERS

Capital Cost Equations: These equations provide the cost of relocating overburden using wheel-type front-end loaders. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by front-end loader.

The base equations assume the following:

1. Haul distance, 500 ft.
2. Rolling resistance, nearly level gradient.
3. Inconsistent operation.
4. Wheel-type loader.

Base Equations:

Equipment operating cost $Y_E = 0.407(X)^{-0.225}$
Labor operating cost .... $Y_L = 13.07(X)^{-0.936}$

Equipment operating costs average 22% parts, 46% fuel and lubrication, and 32% tires. Labor operating costs average 90% operator labor and 10% repair labor.

Distance Factor: If the average haul distance is other than 500 ft, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_D = 0.023(\text{distance})^{0.616}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_G = 0.877e^{[0.046(\text{percent gradient})]}$.

Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor .... $U_e = 1.162(X)^{-0.017}$
Labor factor............ $U_l = 0.989(X)^{0.006}$

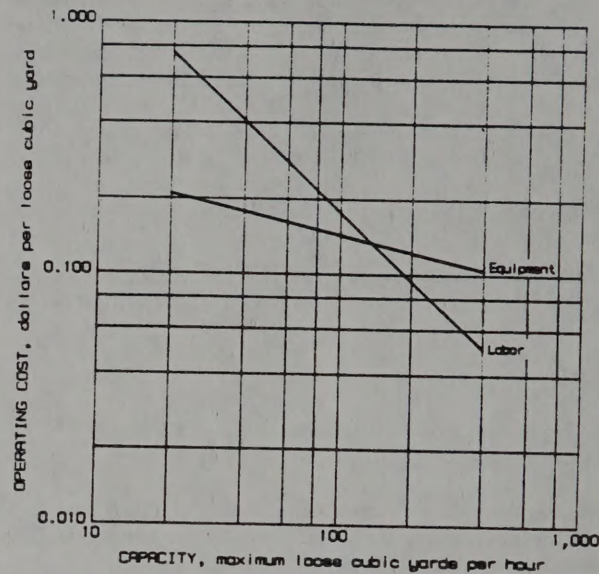
Track-Type Loader Factor: If track-type loaders are used, the following factors must be applied to the total cost obtained from the base equations:

Equipment factor .... $T_e = 1.378$
Labor factor............ $T_l = 1.073$

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e)(T_e) + Y_L(U_l)(T_l)] \times F_D \times F_G.$$

To obtain the overburden removal capital cost, the total cost per loose cubic yard must be multiplied by the total amount of *overburden* handled by front-end loader prior to production. This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Overburden removal capital costs - Front-end loaders

PREPRODUCTION OVERBURDEN REMOVAL—REAR-DUMP TRUCKS

Capital Cost Equations: These equations provide the cost of hauling overburden using rear-dump trucks. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by rear-dump truck.

The base equations assume the following:

1. Haul distance, 2,500 ft.
2. Loader cycles to fill, 4.
3. Efficiency, 50 min/h.
4. Average operator ability.
5. Rolling resistance, 2%, nearly level gradient.

Base Equations:

Equipment operating costs. . $Y_E = 0.602(X)^{-0.296}$
Labor operating cost. . . . . . . $Y_L = 11.34(X)^{-0.891}$

Equipment operating costs consist of 28% parts, 58% fuel and lubrication, and 14% tires. Labor operating costs consist of 82% operator labor and 18% repair labor.

Distance Factor: If average haul distance is other than 2,500 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$$F_D = 0.093(\text{distance})^{0.311}.$$

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$$F_G = 0.907e^{[0.049(\text{percent gradient})]}.$$

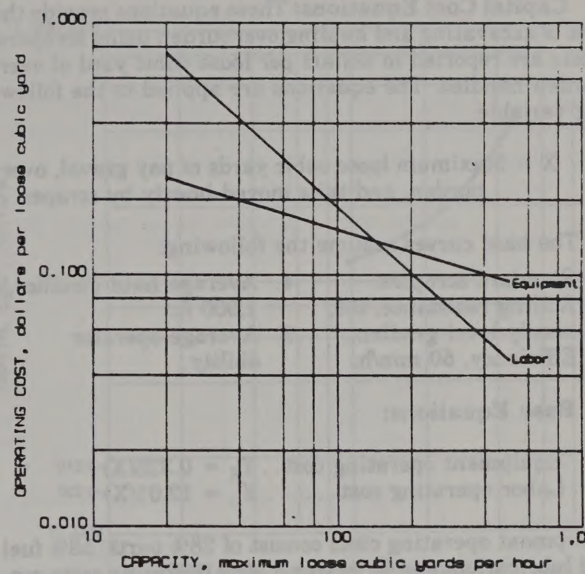
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor. . . . . $U_e = 0.984(X)^{0.016}$
Labor factor. . . . . $U_l = 0.943(X)^{0.021}$

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_D \times F_G.$$

To obtain the overburden removal capital cost, the total cost per loose cubic yard must be multiplied by the total amount of *overburden* handled by truck prior to production. This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Overburden removal capital costs - Rear-dump trucks

PREPRODUCTION OVERBURDEN REMOVAL—SCRAPERS

Capital Cost Equations: These equations provide the cost of excavating and hauling overburden using scrapers. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by scraper.

The base curves assume the following:

1. Standard scrapers.
2. Rolling resistance, 6%, nearly level gradient.
3. Efficiency, 50 min/h.
4. Average haul distance, 1,000 ft.
5. Average operator ability.

Base Equations:

Equipment operating cost. . $Y_E = 0.325(X)^{-0.210}$
Labor operating cost. . . . . . $Y_L = 12.01(X)^{-0.930}$

Equipment operating costs consist of 28% parts, 58% fuel and lubrication, and 14% tires. Labor operating costs consist of 82% operator labor and 18% repair labor.

Distance Factor: If average haul distance is other than 1,000 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$$F_D = 0.01947(\text{distance})^{0.577}.$$

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 6%, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$$F_G = 0.776e^{[0.047(\text{percent gradient})]}.$$

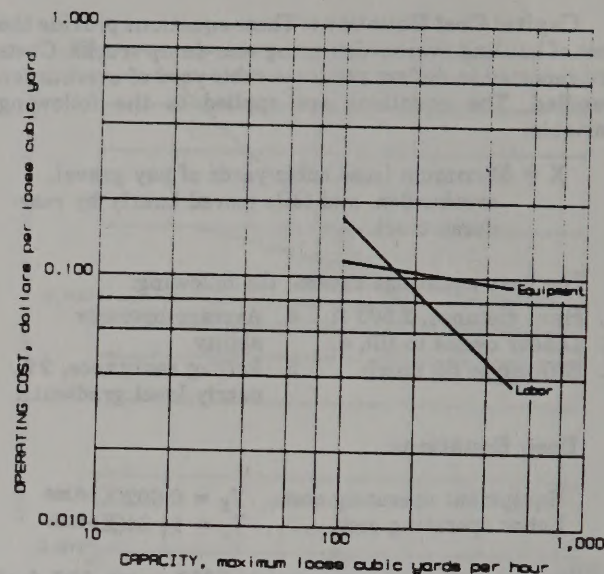
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor. . . . . $U_e = 1.096(X)^{-0.006}$
Labor factor. . . . . $U_l = 0.845(X)^{0.034}$

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_D \times F_G.$$

To obtain the overburden removal capital cost, the total cost per loose cubic yard must be multiplied by the total amount of *overburden* handled by scraper prior to production. This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Overburden removal capital costs - Scrapers

CAPITAL COSTS

MINE EQUIPMENT—BACKHOES

Capital Cost Equation: This equation furnishes the cost of purchasing the appropriate number and size of hydraulic backhoes needed to provide the maximum required production. Costs do not include transportation, sales tax, or discounts. The equation is applied to the following variable:

X = Maximum loose cubic yards of pay gravel moved hourly by backhoe.

The following capacities were used to calculate the base equation:

105 hp	95 to 200 LCY/h	195 hp	250 to 375 LCY/h
135 hp	175 to 275 LCY/h	325 hp	350 to 475 LCY/h

These capacities are based on the following assumptions:

1. Medium digging difficulty.
2. Average operator ability.
3. Swing angle, 60° to 90°.
4. Maximum digging depth, 0% to 50%.
5. No obstructions.

Base Equation:

Equipment capital cost . . . $Y_C = 84{,}132.01e^{(0.00350(X))}$

Equipment capital costs consist entirely of the equipment purchase price.

Digging Depth Factor: If average digging depth is other than 50% of maximum depth obtainable for a particular make of backhoe, the factor obtained from the following equation must be applied to total capital cost:

$F_D = 0.04484(D)^{0.790}$,

where D = percent of maximum digging depth.

Used Equipment Factor: This factor accounts for the reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.386$.

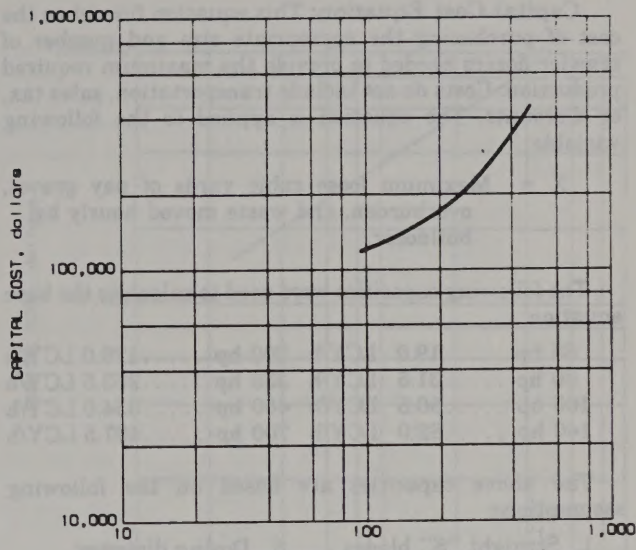
Digging Difficulty Factor: Parameters given in the discussion on site adjustment factors in section 1 should be used to determine if a digging difficulty factor is required. If so, one of the following should be applied to total capital cost:

F_H, easy digging	1.000	F_H, medium-hard digging	1.556
F_H, medium digging	1.330	F_H, hard digging	1.822

Total Cost: Backhoe capital cost is determined by

$Y_C \times F_D \times F_U \times F_H$.

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Mine equipment capital costs - Backhoes

MINE EQUIPMENT—BULLDOZERS

Capital Cost Equation: This equation furnishes the cost of purchasing the appropriate size and number of crawler dozers needed to provide the maximum required production. Costs do not include transportation, sales tax, or discounts. The equation is applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and waste moved hourly by bulldozer.

The following capacities were used to calculate the base equation:

65 hp	.....19.0 LCY/h	200 hp	.....126.0 LCY/h
80 hp	.....31.5 LCY/h	335 hp	.....263.5 LCY/h
105 hp	.....56.5 LCY/h	460 hp	.....334.0 LCY/h
140 hp	.....82.0 LCY/h	700 hp	.....497.5 LCY/h

The above capacities are based on the following assumptions:

1. Straight "S" blades.
2. No ripping.
3. Average operator ability.
4. Cutting distance, 50 ft.
5. Dozing distance, 300 ft.
6. Efficiency, 50 min/h.
7. Even, nearly level gradient.

Base Equation:

Equipment capital cost . . . $Y_C = 3{,}555.96(X)^{0.806}$

Equipment capital costs consist entirely of equipment purchase price.

Distance Factor: If average dozing distance is other than 300 ft, the factor obtained from the following equation must be applied to capital costs. This will correct for the addition or reduction of equipment required to maintain maximum capacity:

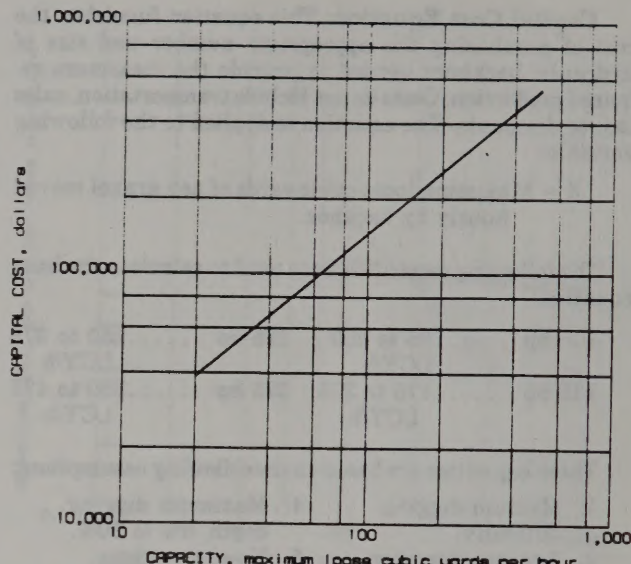
$F_D = 0.01549(\text{distance})^{0.732}$.

Gradient Factor: If the average gradient is other than level, the factor obtained from the following equation must be applied to total capital cost. This will correct for the addition or reduction of equipment required to maintain maximum capacity. (Favorable haul gradients should be entered as negative, uphill haul gradients as positive.)

$F_G = 1.041e^{[0.015(\text{percent gradient})]}$.

Digging Difficulty Factor: Variations from the base digging difficulty will necessitate changes in equipment size to maintain production capacity. Parameters given in the discussion on site adjustment factors in section 1 should be used to determine if a digging difficulty factor is required. If so, one of the following should be applied to total capital cost:

$F_{H,}$ easy digging	..0.863	$F_{H,}$ medium-hard digging	..........1.197
$F_{H,}$ medium digging	..........1.000	$F_{H,}$ hard digging	..1.509



Mine equiment capital costs - Bulldozers

Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.411$.

Total Cost: Bulldozer capital cost is determined by

$Y_C \times F_H \times F_D \times F_G \times F_U$.

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.

MINE EQUIPMENT—DRAGLINES

Capital Cost Equation: This equation furnishes the cost of purchasing the appropriate size dragline needed to provide the maximum required production. Costs do not include transportation, sales tax, or discounts. The equation is applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and waste moved hourly by dragline.

The following capacities were used to calculate the base equation:

84 hp . . . 28 LCY/h	190 hp . . . 94 LCY/h
110 hp . . . 47 LCY/h	263 hp . . . 132 LCY/h
148 hp . . . 66 LCY/h	289 hp . . . 188 LCY/h
170 hp . . . 75 LCY/h	540 hp . . . 264 LCY/h

The above capacities are based on the following assumptions:

1. Bucket efficiency, 0.90.
2. Full hoist.
3. Swing angle, 90°.
4. Average operator ability.

Base Equation:

Equipment capital cost . . . . . $Y_C = 16{,}606.12(X)^{0.678}$

Equipment capital costs consist entirely of the equipment purchase price.

Swing Angle Factor: If the average swing angle is other than 90°, the factor obtained from the following equation must be applied to total capital cost. This factor will compensate for equipment size differences required to obtain the desired maximum capacity:

$F_S = 0.450(\text{swing angle})^{0.180}$.

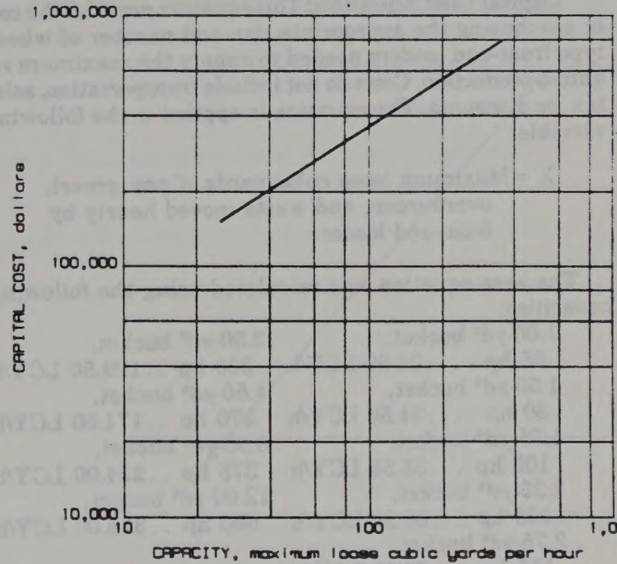
Used Equipment Factor: This factor accounts for the reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.422$.

Total Cost: Dragline capital cost is determined by

$$Y_C \times F_S \times F_U.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Mine equipment capital costs - Draglines

MINE EQUIPMENT—FRONT-END LOADERS

Capital Cost Equation: This equation provides the cost of purchasing the appropriate size and number of wheel-type front-end loaders needed to supply the maximum required production. Costs do not include transportation, sales tax, or discounts. The equation is applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and waste moved hourly by front-end loader.

The base equation was calculated using the following capacities:

1.00-yd³ bucket, 65 hp ....24.00 LCY/h	3.50-yd³ bucket, 200 hp ..129.50 LCY/h
1.50-yd³ bucket, 80 hp ....34.50 LCY/h	4.50-yd³ bucket, 270 hp ..171.00 LCY/h
1.75-yd³ bucket, 105 hp ...38.50 LCY/h	6.50-yd³ bucket, 375 hp ..234.00 LCY/h
2.25-yd³ bucket, 125 hp ...56.25 LCY/h	12.00-yd³ bucket, 690 hp ..348.00 LCY/h
2.75-yd³ bucket, 155 hp ...66.00 LCY/h	

The above capacities are based on the following assumptions:

1. Haul distance, 500 ft.
2. Rolling resistance, 2%, nearly level gradient.
3. Inconsistent operation.
4. Wheel-type loader.
5. Efficiency, 50 min/h.
6. General purpose bucket, heaped.

Base Equation:

Equipment capital cost... $Y_C = 2{,}711.10(X)^{0.896}$

Equipment capital costs consist entirely of the equipment purchase price.

Distance Factor: If the average haul distance is other than 500 ft, the factor obtained from the following equation must be applied to the capital cost. This will correct for the addition or reduction of equipment required to maintain maximum capacity. (If tracked loaders are to be used, the maximum haul distance should not exceed 600 ft.)

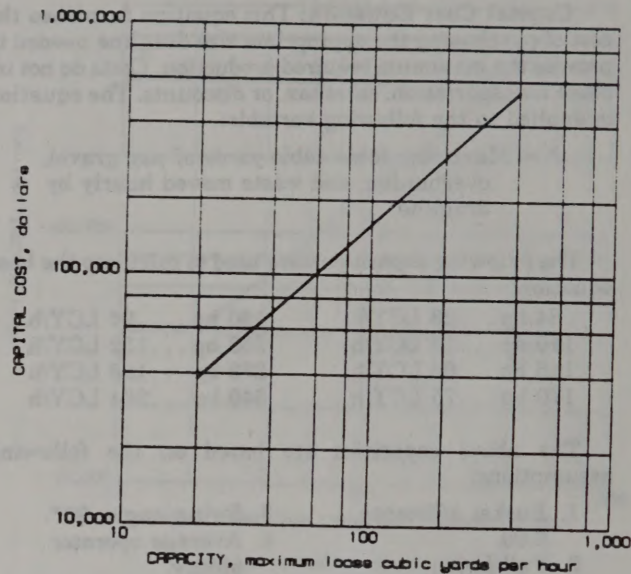
$F_D = 0.033(\text{distance})^{0.552}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to the total capital cost. This will correct for the addition or reduction of equipment required to maintain maximum capacity:

$F_G = 0.888e^{(0.041(\text{percent gradient}))}$.

Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life.

$F_U = 0.386$.



Mine equipment capital costs - Front-end loaders

Track-Type Loader Factor: If track-type loaders are used, the factor obtained from the following equation must be applied to total capital cost. This factor will account for the decrease in production efficiency and the difference in equipment cost:

$F_T = 0.414(X)^{0.272}$.

Total Cost: Front-end loader capital cost is determined by

$$Y_C \times F_D \times F_G \times F_U \times F_T.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.

MINE EQUIPMENT—REAR-DUMP TRUCKS

Capital Cost Equation: This equation furnishes the cost of purchasing the appropriate size and number of diesel rear-dump trucks needed to provide the maximum required production. Costs do not include transportation, sales tax, or discounts. The equation is applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and waste moved hourly by rear-dump truck.

The following capacities were used to calculate the base equation:

3.0-yd^3 truck . . . . . 32.3 LCY/h	12.0-yd^3 truck . . . . . 124.5 LCY/h
5.0-yd^3 truck . . . . . 53.4 LCY/h	16.0-yd^3 truck . . . . . 163.9 LCY/h
6.0-yd^3 truck . . . . . 63.6 LCY/h	22.8-yd^3 truck . . . . . 223.5 LCY/h
8.0-yd^3 truck . . . . . 83.5 LCY/h	34.0-yd^3 truck . . . . . 326.3 LCY/h
10.0-yd^3 truck . . . . . 104.2 LCY/h	47.5-yd^3 truck . . . . . 444.8 LCY/h

The above capacities are based on the following assumptions:

1. Diesel rear-dump trucks.
2. Loader cycles to fill, 4.
3. Haul distance, 2,500 ft.
4. Rolling resistance, 2%, nearly level gradient.

Base Equation:

Equipment capital cost . . . $Y_C = 472.09(X)^{1.139}$

Equipment capital costs consist entirely of the equipment purchase price.

Distance Factor: If the average haul distance is other than 2,500 ft, the factor obtained from the following equation must be applied to capital cost. This will correct for the addition or reduction of equipment required to maintain maximum capacity:

$F_D = 0.06240(\text{distance})^{0.364}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to total capital cost. This will correct for the addition or reduction of equipment required to maintain the maximum capacity. (Favorable haul gradient should be entered as negative, uphill haul grades as positive.)

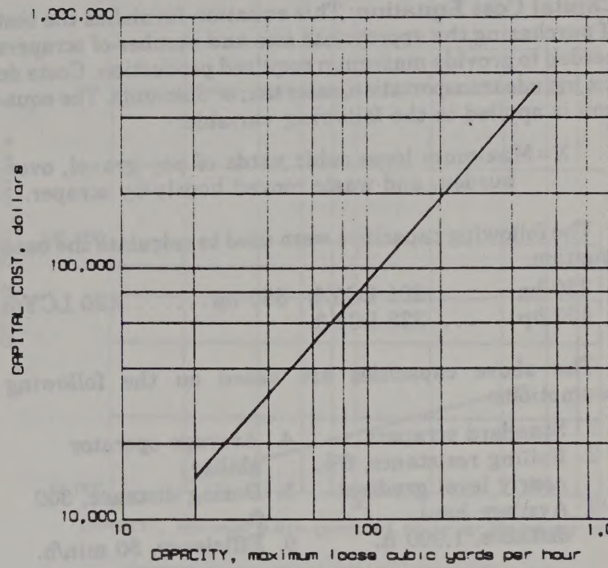
$F_G = 0.896e^{[0.056(\text{percent gradient})]}$.

Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.243$.

Total Cost: Truck capital cost is determined by

$Y_C \times F_D \times F_G \times F_U$.



Mine equipment capital costs - Rear-dump trucks

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.

MINE EQUIPMENT—SCRAPERS

Capital Cost Equation: This equation furnishes the cost of purchasing the appropriate size and number of scrapers needed to provide maximum required production. Costs do not include transportation, sales tax, or discounts. The equation is applied to the following variable:

X=Maximum loose cubic yards of pay gravel, overburden, and waste moved hourly by scraper.

The following capacities were used to calculate the base equation:

330 hp.......201 LCY/h 550 hp.......420 LCY/h
450 hp.......323 LCY/h

The above capacities are based on the following assumptions:

1. Standard scrapers.
2. Rolling resistance, 6%, nearly level gradient.
3. Average haul distance, 1,000 ft.
4. Average operator ability.
5. Dozing distance, 300 ft.
6. Efficiency, 50 min/h.

Base Equation:

Equipment capital cost ..... $Y_C = 1{,}744.42(X)^{0.934}$

Equipment capital costs consist entirely of the equipment purchase price.

Distance Factor: If the haul distance is other than 1,000 ft, the factor obtained from the following equation must be applied to the total capital cost. This will correct for the addition or reduction of equipment required to maintain maximum production capacity:

$F_D = 0.025\ (\text{distance})^{0.539}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 6%, the factor obtained from the following equation must be applied to total capital cost. This will correct for the addition or reduction of equipment required to maintain the maximum production capacity. (Favorable haul gradients are entered as negative, uphill haul gradients as positive.)

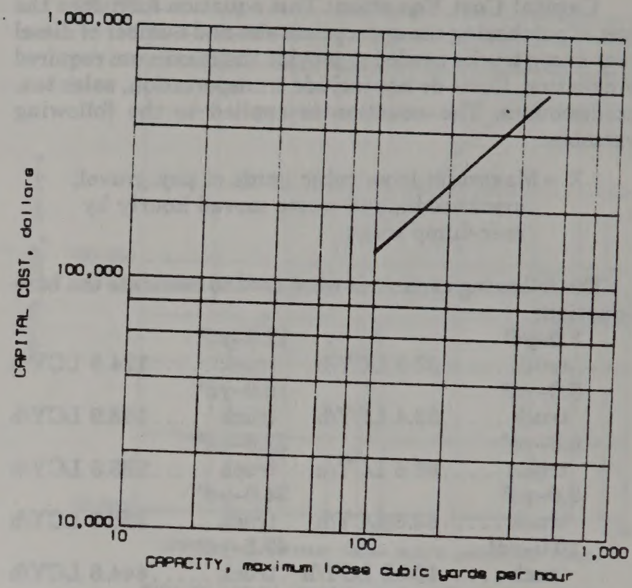
$F_G = 0.776e^{(0.047(\text{percent gradient}))}$.

Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.312$.

Total Cost: Scraper capital cost is determined by $Y_C \times F_D \times F_G \times F_U$.

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Mine equipment capital costs - Scrapers

PROCESSING EQUIPMENT—CONVEYORS

Capital Cost Equation: This equation furnishes the cost of purchasing and installing the appropriate size conveyors needed to meet maximum required production. A separate cost must be calculated for each conveyor in the circuit. The cost includes associated drive motors and electrical hookup. Equipment transportation, sales tax, and discounts are not accounted for. The equation is applied to the following variable:

X = Maximum cubic yards of material moved hourly by conveyor.

The following capacities were used to calculate the base equation:

18-in-wide conveyor ...... 96 yd³/h	30-in-wide conveyor ...... 320 yd³/h
24-in-wide conveyor ...... 192 yd³/h	36-in-wide conveyor ...... 480 yd³/h

Base Equation:

Equipment capital cost ...... $Y_C = 4{,}728.36(X)^{0.287}$

The capital cost consists of 89% equipment purchase price, 8% installation labor, and 3% construction materials.

Length Factor: If the required conveyor length is other than 40 ft, the factor obtained from the following equation must be applied to the calculated capital cost. This factor is valid for conveyors 10 to 100 ft long:

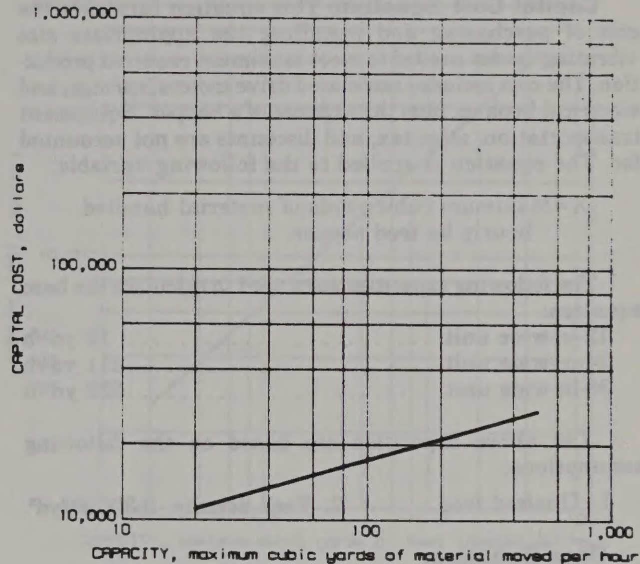
$F_L = 0.304(\text{length})^{0.330}$.

Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.505$.

Total Cost: Conveyor capital cost is determined by $Y_C \times F_L \times F_U$.

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Processing equipment capital costs -Conveyors

PROCESSING EQUIPMENT—FEED HOPPERS

Capital Cost Equation: This equation furnishes the cost of purchasing and installing the appropriate size vibrating feeder needed to meet maximum required production. The cost includes associated drive motors, springs, and electrical hookup, plus the expense of a hopper. Equipment transportation, sales tax, and discounts are not accounted for. The equation is applied to the following variable:

X = Maximum cubic yards of material handled hourly by feed hopper.

The following capacities were used to calculate the base equation:

12-in-wide unit .... 16 yd³/h
24-in-wide unit .... 211 yd³/h
36-in-wide unit .... 522 yd³/h

The above capacities are based on the following assumptions:

1. Unsized feed.
2. Feed density, 2,300 lb/yd³.

Base Equation:

Equipment capital cost .... $Y_C = 458.48(X)^{0.470}$

The capital cost consists of 82% equipment purchase price, 14% construction and installation labor, and 4% steel.

Hopper Factor: In many instances a vibrating feeder may not be required. If a hopper is the only equipment needed, multiply the calculated cost by the factor obtained from the following equation. This factor will account for material and labor required to construct and install a hopper:

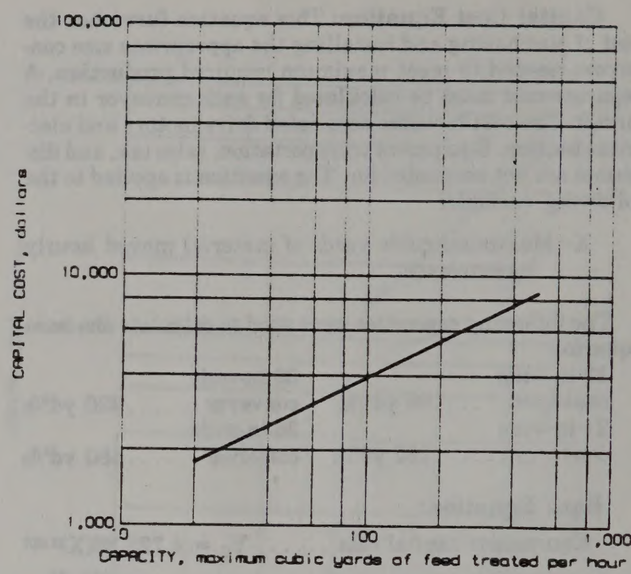
$F_H = 0.078e^{(0.00172(X))}$.

Used Equipment Factor: The factor calculated from the following equation accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.476e^{(0.00036(X))}$.

Total Cost: Feeder capital cost is determined by $Y_C \times F_H \times F_U$.

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Processing equipment capital costs - Feed hoppers

PROCESSING EQUIPMENT—JIG CONCENTRATORS

Capital Cost Equation: This equation furnishes the cost of purchasing and installing the appropriate size and number of jigs needed to meet maximum required production. The cost includes associated drive motors, piping, and electrical hookup. Equipment transportation, sales tax, and discounts are not accounted for. The equation is applied to the following variable:

X = Maximum cubic yards of feed handled hourly by jig concentrators.

The following capacities were used to calculate the base equation:

12- by 12-in simplex ..... 0.617 yd³/h	36- by 36-in triplex ... 16.659 yd³/h
26- by 26-in simplex ..... 2.896 yd³/h	42- by 42-in triplex ... 22.675 yd³/h
36- by 36-in duplex ...... 11.106 yd³/h	

The above capacities are based on the following assumptions:

1. Cleaner service.
2. Hourly capacity, 0.617 yd^3/ft^2.
3. Feed solids, 3,400 lb/yd^3.
4. Slurry density, 40% solids.
5. Gravity feed.

Base Equation:

Equipment capital cost ...... $Y_C = 6{,}403.82(X)^{0.595}$

The capital cost consists of 62% equipment purchase price, 12% construction labor and installation, and 26% construction materials.

Rougher-Coarse Factor: If jigs are to be used for rougher service, or a coarse feed, higher productivity will be realized. To account for the reduction in equipment required to maintain production, the calculated capital cost must be multiplied by the following factor:

$F_R = 0.531.$

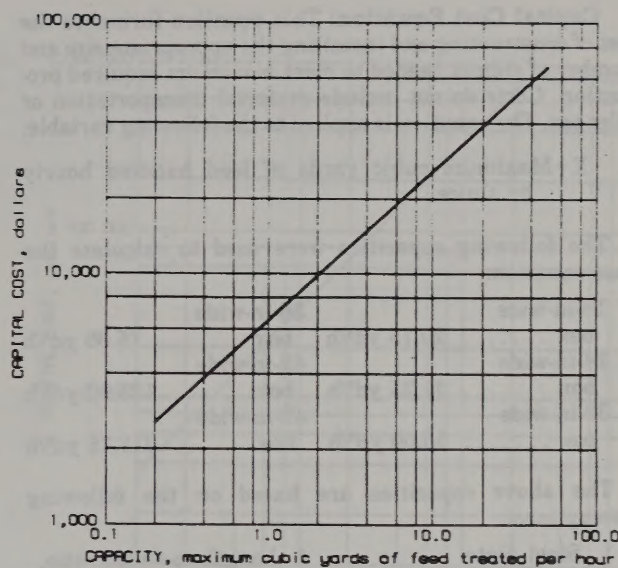
Used Equipment Factor: This factor accounts for the reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.697.$

Total Cost: Jig concentrator capital cost is determined by

$$Y_C \times F_R \times F_U.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Processing equipment capital costs - Jig concentrators

PROCESSING EQUIPMENT—SLUICES

Capital Cost Equation: This equation furnishes the cost of constructing and installing the appropriate size and number of sluices needed to meet maximum required production. Costs do not include material transportation or sales tax. The equation is applied to the following variable:

X=Maximum cubic yards of feed handled hourly by sluice.

The following capacities were used to calculate the base equation:

18-in-wide box	20.75 yd³/h	36-in-wide box	75.00 yd³/h
24-in-wide box	31.25 yd³/h	42-in-wide box	125.00 yd³/h
30-in-wide box	50.00 yd³/h	48-in-wide box	218.75 yd³/h

The above capacities are based on the following assumptions:

1. Steel plate construction.
2. Angle-iron riffles.
3. Feed solids, 3,400 lb/yd³.
4. Length-to-width ratio, 4:1
5. Gravity feed.

Base Equation:

Equipment capital cost . . . . $Y_C = 113.57(X)^{0.567}$

The capital cost consists of 61% construction and installation labor, and 39% construction materials.

Wood Construction Factor: If sluices are to be made of wood rather than steel, the following factor will account for reduced material and construction costs:

$F_W = 0.499(X)^{-0.023}$.

Length Factor: This factor will account for changes in the desired length of the sluice. The factor obtained from the following equation must be applied to capital cost:

$F_L = 1.001(L)^{0.753}$,

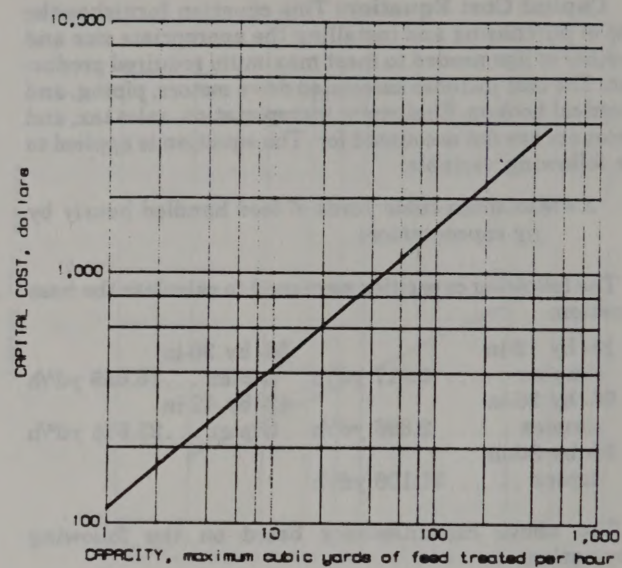
where L = desired length divided by length assumed for the base calculation (width × 4.0).

Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life.

$F_U = 0.574$.

Total Cost: Sluice capital cost is determined by $Y_C \times F_W \times F_L \times F_U$.

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Processing equipment capital costs - Sluices

PROCESSING EQUIPMENT—SPIRAL CONCENTRATORS

Capital Cost Equation: This equation furnishes the cost of purchasing and installing the appropriate number of spirals needed to meet maximum required production. Cost of slurry splitters, fittings, and pipe are all included. Costs do not include transportation, sales tax, or discounts. The equation is applied to the following variable:

X = Maximum cubic yards of feed handled hourly by spiral concentrator.

The following capacities were used to calculate the base equation:

2 starts .	2 yd^3/h	50 starts	50 yd^3/h
10 starts .	10 yd^3/h	100 starts	100 yd^3/h

The above capacities are based on the following assumptions:

1. Rougher service.
2. Solids per start, 1.75 st/h.
3. Feed solids, 3,400 lb/yd^3.
4. Slurry density, 10% solids.
5. Gravity feed.

Base Equation:

Equipment capital cost... $Y_C = 3{,}357.70(X)^{0.999}$

The capital cost consists of 71% equipment purchase price, 13% construction labor and installation, and 16% construction materials.

Cleaner-Scavenger Service Factor: If spirals are to be used for cleaner or scavenger functions, unit capacity will decrease. To account for additional equipment needed to maintain production, calculated capital cost must be multiplied by the following factor:

$F_C = 2.333.$

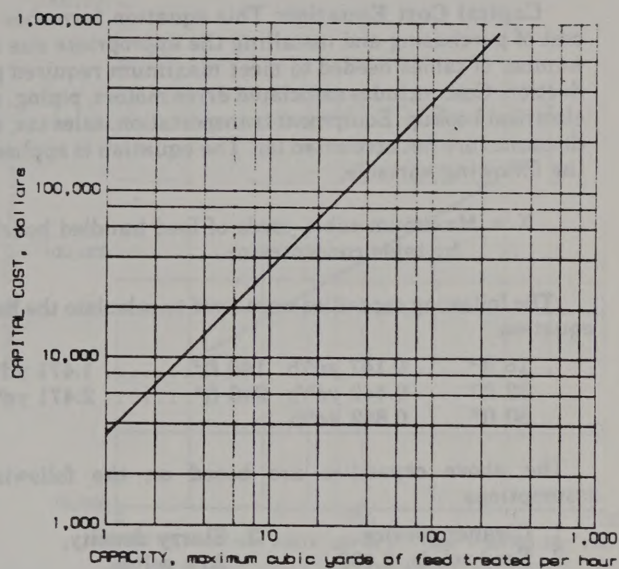
Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.654.$

Total Cost: Spiral concentrator capital cost is determined by

$$Y_C \times F_C \times F_U.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Processing equipment capital costs - Spiral concentrators

PROCESSING EQUIPMENT—TABLE CONCENTRATORS

Capital Cost Equation: This equation furnishes the cost of purchasing and installing the appropriate size and number of tables needed to meet maximum required production. Cost includes associated drive motors, piping, and electrical hookup. Equipment transportation, sales tax, and discounts are not accounted for. The equation is applied to the following variable:

X = Maximum cubic yards of feed handled hourly by table concentrator.

The following capacities were used to calculate the base equation:

18 ft² .... 0.147 yd³/h	140 ft² .... 1.471 yd³/h
32 ft² .... 0.442 yd³/h	240 ft² .... 2.471 yd³/h
80 ft² .... 0.882 yd³/h	

The above capacities are based on the following assumptions.

1. Cleaner service.
2. Feed solids, 3,400 lb/yd³.
3. Slurry density, 25% solids.
4. Gravity feed.

Base Equation:

Equipment capital cost... $Y_C = 20{,}598.06(X)^{0.643}$

The capital cost consists of 62% equipment purchase price, 12% construction labor and installation, and 26% construction materials.

Rougher-Coarse Factor: If tables are to be used for rougher service, or a coarse feed, higher productivity will be realized. To account for reduction in equipment required to maintain production, the calculated capital cost must be multiplied by the following factor:

$F_R = 0.568.$

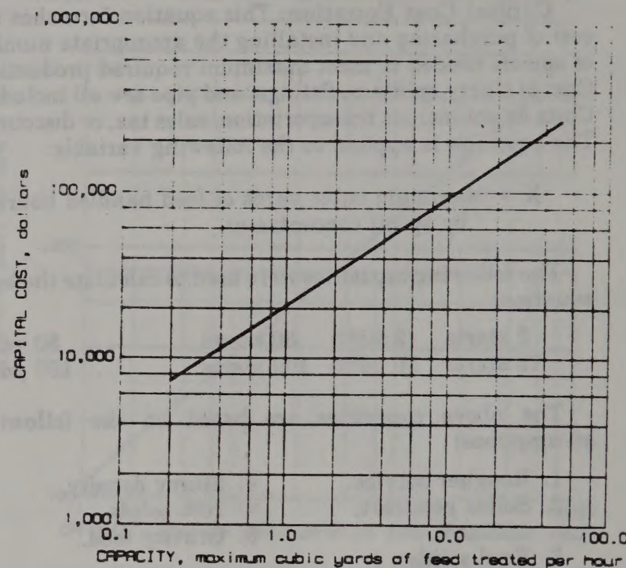
Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.596.$

Total Cost: Table concentrator capital cost is determined by

$$Y_C \times F_R \times F_U.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Processing equipment capital costs - Table concentrators

PROCESSING EQUIPMENT—TROMMELS

Capital Cost Equation: This equation furnishes the cost of purchasing and installing the appropriate size trommels needed to meet maximum required production. Cost includes associated drive motors, piping, and electrical hookup. Equipment transportation, sales tax, and discounts are not accounted for. The equation is applied to the following variable:

X = Maximum cubic yards of feed handled hourly by trommels.

The following capacities were used to calculate the base equation:

3.0-ft diam . . .	40 yd^3/h.	5.0-ft diam . . .	250 yd^3/h.
3.5-ft diam . . .	50 yd^3/h.	5.5-ft diam . . .	300 yd^3/h.
4.0-ft diam . . .	85 yd^3/h.	7.0-ft diam . . .	500 yd^3/h.
4.5-ft diam . . .	150 yd^3/h.		

The above capacities are based on the following assumptions:

1. Trommels are sectioned for scrubbing and sizing.
2. Gravity feed.
3. Feed density, 2,300 lb/yd^3.

Base Equation:

Equipment capital cost . . . $Y_C = 7{,}176.21(X)^{0.559}$

The capital cost consists of 64% equipment purchase price, 26% construction and installation labor, and 10% construction materials.

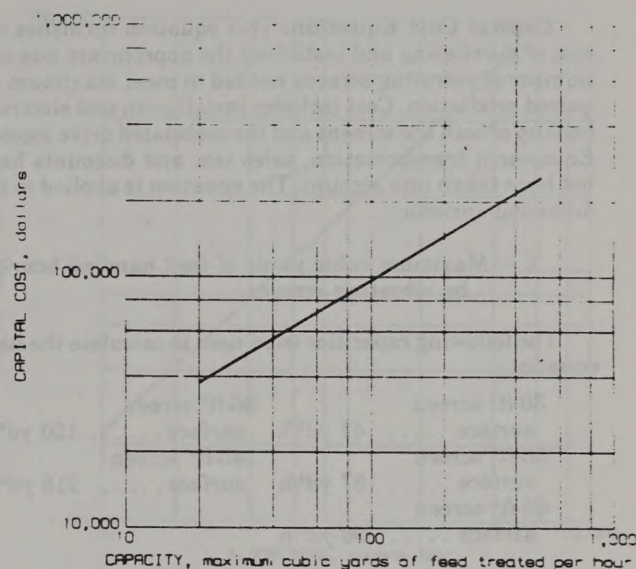
Used Equipment Factor: This factor accounts for the reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.516.$

Total Cost: Trommel capital cost is determined by

$$Y_C \times F_U.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Processing equipment capital costs - Trommels

PROCESSING EQUIPMENT—VIBRATING SCREENS

Capital Cost Equation: This equation furnishes the cost of purchasing and installing the appropriate size and number of vibrating screens needed to meet maximum required production. Cost includes installation and electrical hookup of both the screens and the associated drive motors. Equipment transportation, sales tax, and discounts have not been taken into account. The equation is applied to the following variable:

X = Maximum cubic yards of feed handled hourly by vibrating screens.

The following capacities were used to calculate the base equation:

30-ft^2 screen surface	47 yd^3/h	96-ft^2 screen surface	150 yd^3/h
56-ft^2 screen surface	87 yd^3/h	140-ft^2 screen surface	218 yd^3/h
60-ft^2 screen surface	93 yd^3/h		

The above capacities are based on the following assumptions:

1. An average of 0.624 ft^2 of screen is required for every cubic yard of hourly capacity.
2. Feed solids, 3,120 lb/yd^3.
3. Gravity feed.

Base Equation:

Equipment capital cost... $Y_C = 1{,}870.20(X)^{0.631}$

The capital cost consists of 75% equipment purchase price, 10% construction and installation labor, and 15% construction materials.

Capacity Factor: If anticipated screen capacity is other than 0.624 ft^2/yd^3 of hourly feed capacity, the calculated capital cost must be multiplied by the following factor. This will account for the increase or reduction in equipment size required to maintain production:

$F_C = 1.322(C)^{0.629}$,

where C = anticipated capacity in square feet per cubic yard of hourly feed.

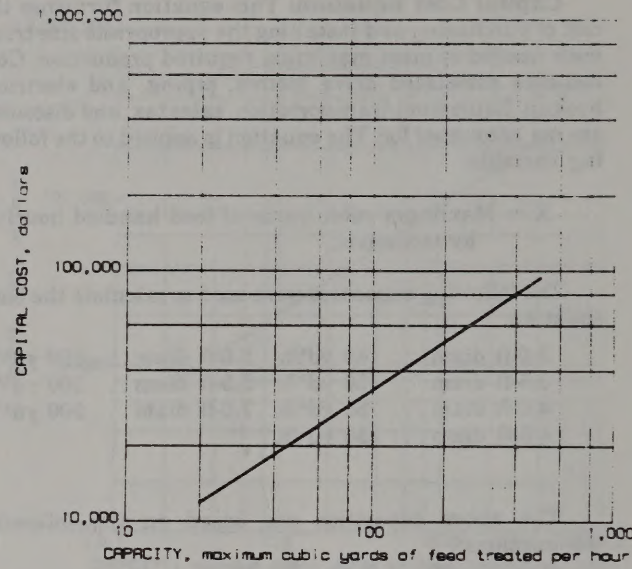
Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.565$.

Total Cost: Vibrating screen capital cost is determined by

$$Y_C \times F_C \times F_U.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Processing equipment capital costs - Vibrating screens

SUPPLEMENTAL—BUILDINGS

Capital Cost Equation: This equation provides the cost of materials and construction for any buildings needed at the site. These may include storage sheds, shops, or mill buildings. Costs do not include sales tax, material transportation, or discounts. A separate cost must be calculated for each building, and the equation is applied to the following variable:

X = Estimated floor area, in square feet.

Building costs are based on the following assumptions:

1. Average quality temporary structures.
2. Steel frame with metal siding and roofing.
3. Concrete perimeter foundations with wood floors.
4. Electricity and lighting provided.

Base Equation:

Capital cost .... $Y_C = 34.09(X)^{0.907}$

The capital cost consists of 34% construction labor, 41% construction materials, and 25% equipment.

Cement Floor Factor: If a cement floor is required, the cost calculated from the base equation must be multiplied by the factor obtained from the following equation:

$F_C = 1.035(X)^{0.008}$.

Plumbing Factor: If plumbing is required, the following factor must be applied to the total capital cost:

$F_P = 1.013(X)^{0.002}$.

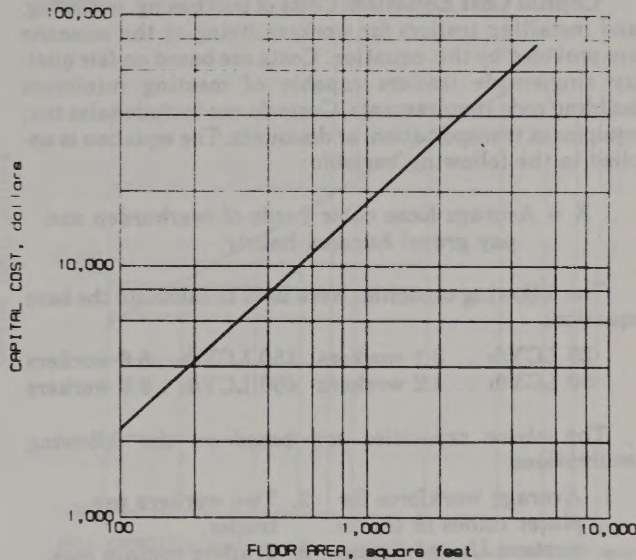
Foundation Factor: If a concrete foundation and wood floor are not needed, multiply the capital cost by the factor obtained from the following equation. This will account for the cost of wood blocks and sills for the foundation:

$F_F = 0.640(X)^{0.026}$.

Total Cost: Building capital cost is determined by

$$Y_C \times F_C \times F_P \times F_F.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Supplemental capital costs - Buildings

SUPPLEMENTAL—EMPLOYEE HOUSING

Capital Cost Equation: Costs of purchasing, outfitting, and installing trailers for workers living at the minesite are provided by this equation. Costs are based on fair quality single-wide trailers capable of meeting minimum building code requirements. Costs do not include sales tax, equipment transportation, or discounts. The equation is applied to the following variable:

X = Average loose cubic yards of overburden and pay gravel handled hourly.

The following capacities were used to calculate the base equation:

25 LCY/h .. 3.1 workers 150 LCY/h . 6.6 workers
50 LCY/h .. 4.2 workers 400 LCY/h . 9.9 workers

The above capacities are based on the following assumptions:

1. Average workforce for placer mines in the western United States (including Alaska).
2. Two workers per trailer.
3. Trailers contain cooking facilities.

Base Equation:

Capital cost .... $Y_C = 7{,}002.51(X)^{0.418}$

The capital cost consists of 90% equipment purchase price, 7% construction and installation labor, and 3% construction materials.

Used Equipment Factor: This factor accounts for the reduced expense of purchasing used trailers. The adjusted cost is obtained by multiplying the calculated capital cost by the following factor:

$F_U = 0.631.$

Workforce Factor: The equation used to compute labor for capital cost estimation is:

$\text{Workforce} = 0.822(X)^{0.415}.$

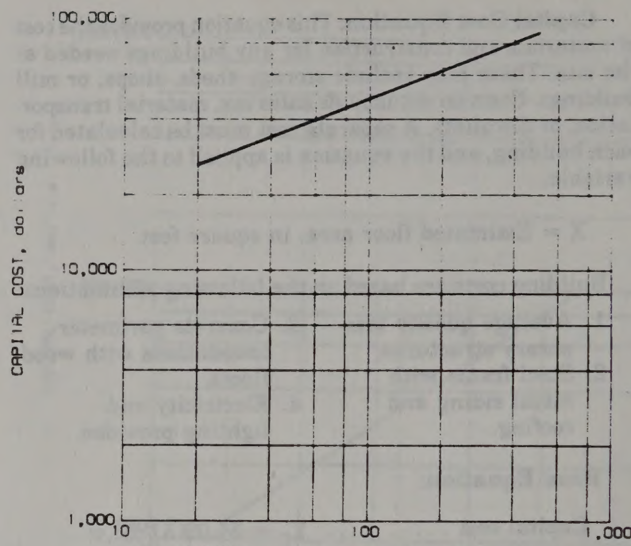
If the workforce for the operation under evaluation is known, and is different than that calculated from the above equation, the correct capital cost may be obtained from the following equation:

$Y_C = (\text{Number of workers}) \times 8{,}608.18.$

Total Cost: Employee housing capital cost is determined by

$$Y_C \times F_U.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Supplemental capital costs - Employee housing

SUPPLEMENTAL—GENERATORS

Capital Cost Equation: This equation provides the cost of purchasing and installing the appropriate size generator required to meet maximum production. Cost includes installation and connection through the fuse box, and allows for mill, mine, camp, and ancillary function power consumption. Costs do not include equipment transportation, sales, tax, or discounts. The equation is applied to the following variable:

X = Maximum cubic yards of feed handled per hour.

The following capacities were used to calculate the base equation:

10-kW generator	10 yd^3/h	75-kW generator	125 yd^3/h
30-kW generator	40 yd^3/h	125-kW generator	200 yd^3/h
45-kW generator	75 yd^3/h	250-kW generator	400 yd^3/h

The above capacities are based on the assumption that 0.57 kW is needed for every cubic yard of mill capacity. This is average for a mine with a basic plant containing trommels, conveyors, mechanical gravity separation devices (jigs or tables), and other necessary ancillary equipment. In all cases, a slightly higher rated generator has been selected for costing purposes to account for demand surges and miscellaneous electrical consumption, such as camp electricity. A factor is provided below for operations with power consumption rates other than 0.57 kW/yd^3.

Base Equation:

Equipment capital cost . . . $Y_C = 1{,}382.65(X)^{0.604}$

The capital cost consists of 75% equipment purchase price, 19% construction and installation labor, and 6% construction materials.

Alternate Power Consumption Factor: If anticipated power consumption rate is other than 0.57 kW/yd^3 mill capacity, the capital cost must be multiplied by the factor obtained from the following equation:

$$F_P = 1.365(P)^{0.618},$$

where P = anticipated power consumption rate.

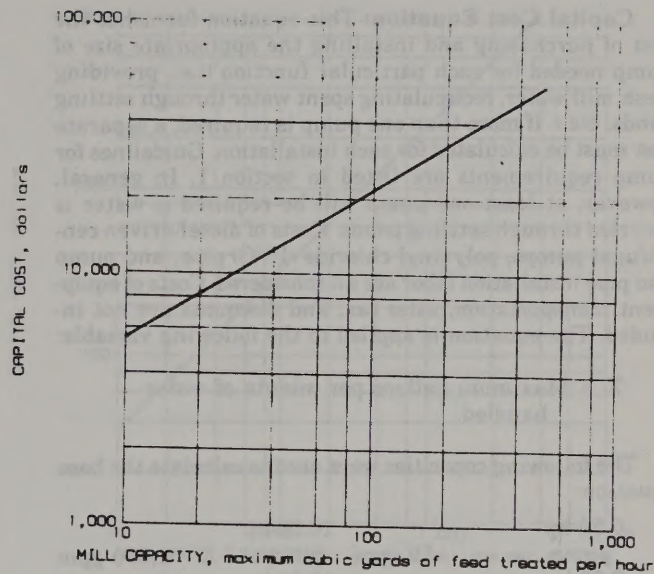
Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$$F_U = 0.481.$$

Total Cost: Generator capital cost is determined by

$$Y_C \times F_P \times F_U.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Supplemental capital costs - Generators

SUPPLEMENTAL—PUMPS

Capital Cost Equation: This equation furnishes the cost of purchasing and installing the appropriate size of pump needed for each particular function (i.e., providing fresh mill water, recirculating spent water through settling ponds, etc.). If more than one pump is required, a separate cost must be calculated for each installation. Guidelines for pump requirements are listed in section 1. In general, however, at least one pump will be required if water is recycled through settling ponds. Costs of diesel-driven centrifugal pumps, polyvinyl chloride (PVC) pipe, and pump and pipe installation labor are all considered. Costs of equipment transportation, sales tax, and discounts are not included. The equation is applied to the following variable:

X = Maximum gallons per minute of water handled.

The following capacities were used to calculate the base equation:

0.50-hp pump	50 gpm	10.50-hp pump	1,000 gpm
2.00-hp pump	200 gpm	18.50-hp pump	1,750 gpm
5.25-hp pump	500 gpm	37.00-hp pump	3,500 gpm

The above capacities are based on the following assumptions:

1. Total head of 25 ft.
2. Diesel-powered pumps.
3. Abrasion-resistant steel construction.
4. Total engine-pump efficiency of 60%.

Base Equation:

Equipment capital cost . . . $Y_C = 63.909(X)^{0.618}$

The capital cost consists of 70% equipment purchase price, 22% construction materials, and 8% construction and installation labor.

Head Factor: If total pumping head is other than 25 ft, the factor calculated from the following equation will correct for changes in pump size requirements. The product of this factor and the original cost will provide the appropriate figure:

$F_H = 0.125(H)^{0.637}$,

where H = total pumping head.

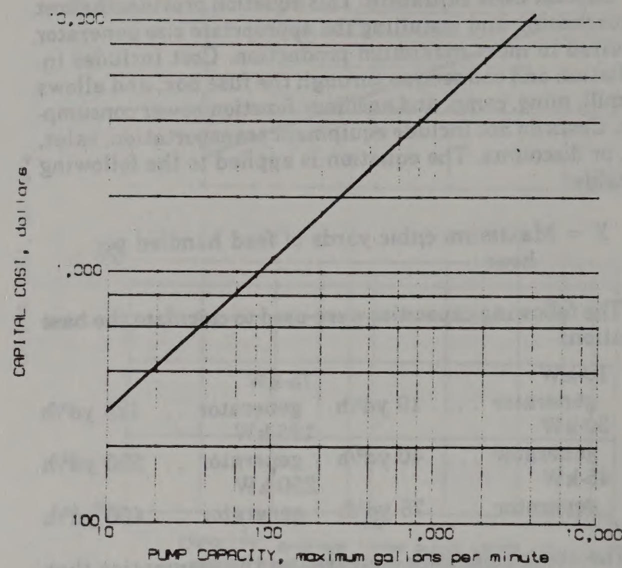
Used Equipment Factor: This factor accounts for reduced capital expenditure of purchasing equipment having over 10,000 h of previous service life:

$F_U = 0.615$.

Total Cost: Pump capital cost is determined by

$Y_C \times F_H \times F_U$.

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Supplementa. capital costs - Pumps

SUPPLEMENTAL—SETTLING PONDS

Capital Cost Equation: This equation furnishes the cost of settling ponds for waste-water treatment. Costs of labor and equipment operation for site selection, size determination, rough surveying, excavation, ditching, grading, and placement of sized gravel are all included. The equation is applied to the following variable:

X = Maximum mill water consumption, in gallons per minute.

If the water consumption rate is not known, one can be estimated from the following equation:

$$X = 94.089(Y)^{0.546},$$

where Y = maximum cubic yards of mill feed handled per hour.

The following capacities were used to calculate the base equation:

400 gpm .... $1,426\text{-yd}^3$ liquid capacity.
600 gpm .... $2,139\text{-yd}^3$ liquid capacity
900 gpm .... $3,208\text{-yd}^3$ liquid capacity
1,400 gpm ... $4,991\text{-yd}^3$ liquid capacity

The above capacities are based on the following assumptions:

1. Pond located in mined-out area.
2. Excavated by bulldozer.
3. Capable of holding 12 h of waste water produced by mill.
4. Based on jig plant water consumption rate.

Base Equation:

Capital cost... $Y_C = 3.982(X)^{0.952}$

The capital cost consists of 75% construction labor, 13% fuel and lubrication, and 12% equipment parts.

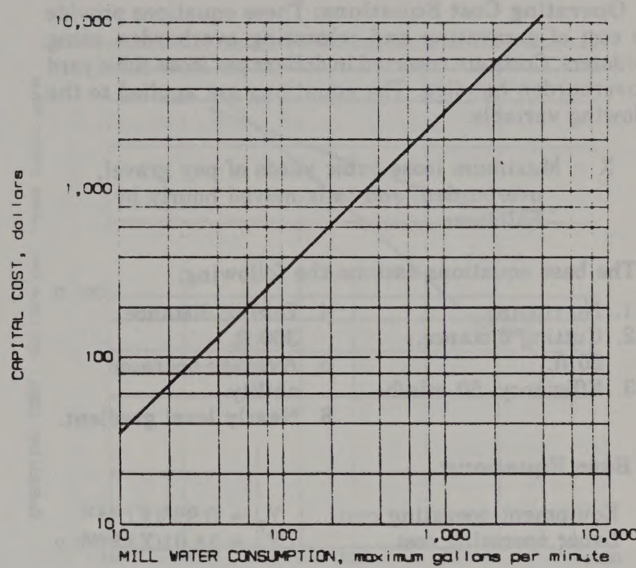
Liner Factor: In order to meet water quality standards, some settling ponds must be lined with an impervious material. If such a liner is required, total capital cost must be multiplied by the factor calculated from the following equation: This factor covers cost of the liner and associated installation labor:

$$F_L = 27.968(X)^{-0.314}.$$

Total Cost: Settling pond capital cost is determined by

$$Y_C \times F_L.$$

This product is subsequently entered in the appropriate row of the tabulation shown in figure 5 for final capital cost calculation.



Supplemental capital costs - Settling ponds

OVERBURDEN REMOVAL—BULLDOZERS

Operating Cost Equations: These equations provide the cost of excavating and relocating overburden using bulldozers. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by bulldozer.

The base equations assume the following:

1. No ripping.
2. Cutting distance, 50 ft.
3. Efficiency, 50 min/h.
4. Dozing distance, 300 ft.
5. Average operator ability.
6. Nearly level gradient.

Base Equations:

Equipment operating cost . . . $Y_E = 0.993(X)^{-0.430}$
Labor operating cost . . . . $Y_L = 14.01(X)^{-0.945}$

Equipment operating costs average 47% parts and 53% fuel and lubrication. Labor operating costs average 86% operator labor and 14% repair labor.

Distance Factor: If the average dozing distance is other than 300 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_D = 0.00581(\text{distance})^{0.904}$.

Gradient Factor: If the average gradient is other than level, the factor obtained from the following equation must be appled to the total cost per loose cubic yard:

$F_G = 1.041e^{[0.015(\text{percent gradient})]}$.

Ripping Factor: If ripping is required; total operating cost must be multiplied by the following factor. This will account for the reduced productivity associated with ripping:

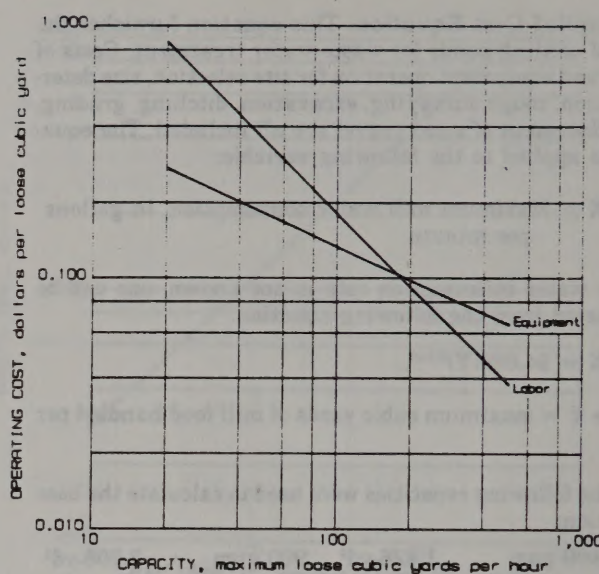
$F_R = 1.595$.

Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by the factors obtained from the following equations:

Equipment factor . . . . $U_e = 1.206(X)^{-0.013}$
Labor factor . . . . $U_l = 0.967(X)^{0.015}$

Digging Difficulty Factor: Parameters given in the discussion on site adjustment factors in section 1 should be used to determine if a digging difficulty factor is required. If so, one of the following should be applied to total cost per loose cubic yard:

F_H, easy digging . 0.830
F_H, medium digging 1.000
F_H, medium-hard digging 1.250
F_H, hard digging . 1.670



Overburden removal operating costs - Bulldozers

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_D \times F_G \times F_H \times F_R.$$

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *overburden* handled by bulldozer. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.

OVERBURDEN REMOVAL—DRAGLINES

Operating Cost Equations: These equations provide the cost of excavating overburden using draglines. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by dragline.

The base curves assume the following:

1. Bucket efficiency, 0.90.
2. Full hoist.
3. Swing angle, 90°.
4. Average operator ability.

Base Equations:

Equipment operating cost . . . $Y_E = 1.984(X)^{-0.390}$
Labor operating cost . . . . $Y_L = 12.19(X)^{-0.888}$

Equipment operating costs consist of 67% parts and 33% fuel and lubrication. Labor operating costs consist of 78% operator labor and 22% repair labor.

Swing Angle Factor: If average swing angle is other than 90°, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_S = 0.304(\text{swing angle})^{0.269}$.

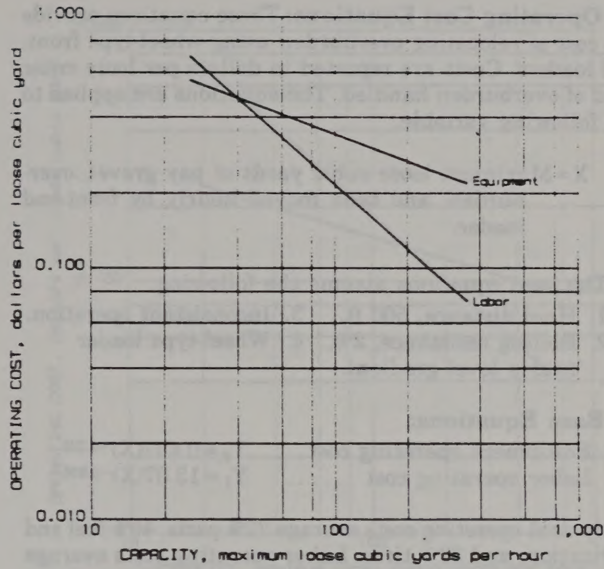
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by the factors obtained from the following equations:

Equipment factor . . . . $U_e = 1.162(X)^{-0.017}$
Labor factor . . . . $U_l = 0.989(X)^{0.006}$

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_S.$$

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *overburden* handled by dragline. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Overburden removal operating costs - Draglines

OVERBURDEN REMOVAL—FRONT-END LOADERS

Operating Cost Equations: These equations provide the cost of relocating overburden using wheel-type front-end loaders. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by front-end loader.

The base equations assume the following:

1. Haul distance, 500 ft.
2. Rolling resistance, 2%, nearly level gradient.
3. Inconsistent operation.
4. Wheel-type loader

Base Equations:

Equipment operating cost . . . $Y_E = 0.407(X)^{-0.225}$
Labor operating cost . . . . $Y_L = 13.07(X)^{-0.936}$

Equipment operating costs average 22% parts, 46% fuel and lubrication, and 32% tires. Labor operating costs average 90% operator labor and 10% repair labor.

Distance Factor: If average haul distance is other than 500 ft, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_D = 0.023(\text{distance})^{0.616}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_G = 0.877e^{[0.046(\text{percent gradient})]}$.

Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by the factors obtained from the following equations:

Equipment factor . . . . $U_e = 1.162(X)^{-0.017}$
Labor factor . . . . $U_l = 0.989(X)^{0.006}$

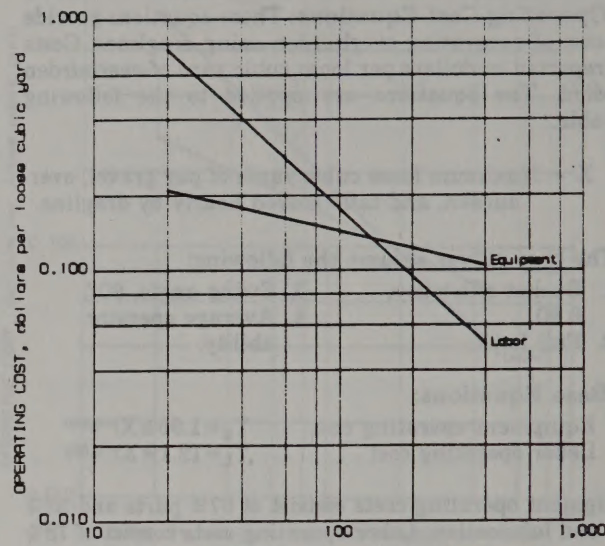
Track-Type Loader Factor: If track-type loaders are used, the following factors must be applied to the total cost obtained from the base equations:

Equipment factor . . . . $T_e = 1.378$
Labor factor . . . . $T_l = 1.073$

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e)(T_e) + Y_L(U_l)(T_l)] \times F_D \times F_G.$$

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *overburden* handled by dragline. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Overburden removal operating costs - Front-end loaders

OVERBURDEN REMOVAL—REAR-DUMP TRUCKS

Operating Cost Equations: These equations provide the cost of hauling overburden using rear-dump trucks. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X=Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by rear dump truck.

The base equations assume the following:

1. Haul distance, 2,500 ft.
2. Loader cycles to fill, 4.
3. Efficiency, 50 min/h.
4. Average operator ability.
5. Nearly level gradient.

Base Equations:

Equipment operating cost . . . $Y_E=0.602(X)^{-0.296}$
Labor operating cost . . . . $Y_L=11.34(X)^{-0.891}$

Equipment operating costs consist of 28% parts, 58% fuel and lubrication, and 14% tires. Labor operating costs consist of 82% operator labor and 18% repair labor.

Distance Factor: If average haul distance is other than 2,500 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_D=0.093(\text{distance})^{0.311}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_G=0.907e^{[0.049(\text{percent gradient})]}$.

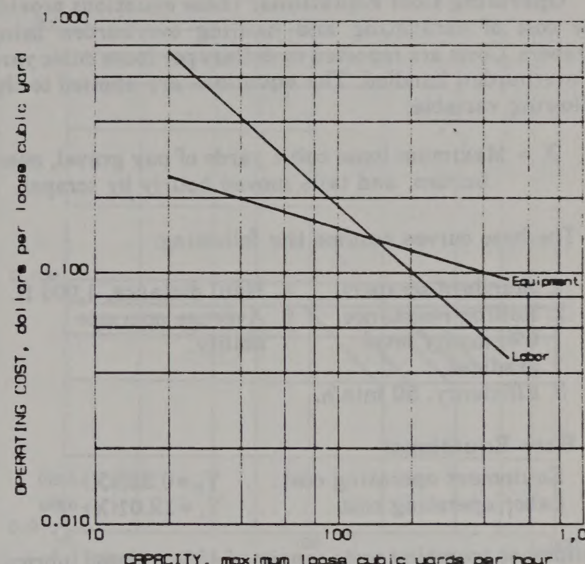
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by the factors obtained from the following equations:

Equipment factor . . . . $U_e=0.984(X)^{-0.016}$
Labor factor . . . . $U_l=0.943(X)^{0.021}$

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e+Y_L(U_l)] \times F_D \times F_G.$$

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *overburden* handled by truck. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Overburden removal operating costs - Rear-dump trucks

OVERBURDEN REMOVAL—SCRAPERS

Operating Cost Equations: These equations provide the cost of excavating and hauling overburden using scrapers. Costs are reported in dollars per loose cubic yard of overburden handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by scraper.

The base curves assume the following:

1. Standard scrapers.
2. Rolling resistance, 6%, nearly level gradient.
3. Efficiency, 50 min/h.
4. Haul distance, 1,000 ft.
5. Average operator ability.

Base Equations:

Equipment operating cost . . . $Y_E = 0.325(X)^{-0.210}$
Labor operating cost . . . . $Y_L = 12.01(X)^{-0.930}$

Equipment operating costs consist of 48% fuel and lubrication, 34% tires, and 18% parts. Labor operating costs consist of 88% operator labor and 12% repair labor.

Distance Factor: If average haul distance is other than 1,000 ft, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_D = 0.01947(\text{distance})^{0.577}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 6%, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_G = 0.776e^{(0.047(\text{percent gradient}))}$.

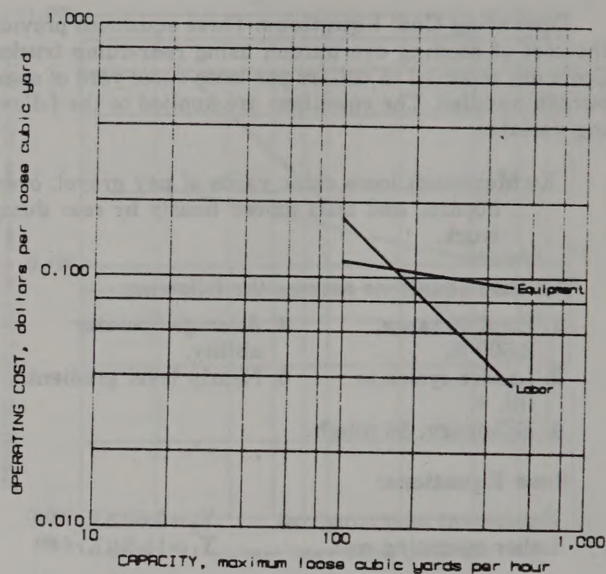
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by the factors obtained from the following equations:

Equipment factor . . . . $U_e = 1.096(X)^{-0.006}$
Labor factor . . . . $U_l = 0.845(X)^{0.034}$

Total Cost: Cost per loose cubic yard of overburden is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_D \times F_G.$$

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *overburden* handled by scraper. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Overburden removal operating costs - Scrapers

MINING—BACKHOES

Operating Cost Equations: These equations provide the cost of excavating pay gravel using backhoes. Costs are reported in dollars per loose cubic yard of pay gravel handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel moved hourly by backhoe.

The base equations assume the following:

1. Easy digging difficulty.
2. Swing angle, 60° to 90°.
3. Up to 50% of maximum digging depth.
4. Average operator ability.
5. No obstructions (boulders, tree roots, etc.).

Base Equations:

95–200 LCY/h:

Equipment operating cost . . . $Y_E=8.360(X)^{-1.019}$
Labor operating cost . . . . $Y_L=17.53(X)^{-1.009}$

175–275 LCY/h:

Equipment operating cost . . . $Y_E=11.44(X)^{-1.021}$
Labor operating cost . . . . $Y_L=17.25(X)^{-1.000}$

250–375 LCY/h:

Equipment operating cost . . . $Y_E=15.17(X)^{-1.003}$
Labor operating cost . . . . $Y_L=19.97(X)^{-1.017}$

350–475 LCY/h:

Equipment operating cost . . . $Y_E=22.59(X)^{-0.995}$
Labor operating cost . . . . $Y_L=16.55(X)^{-0.977}$

Equipment operating costs consist of 38% parts and 62% fuel and lubrication. Labor operating costs consist of 88% operator labor and 12% repair labor.

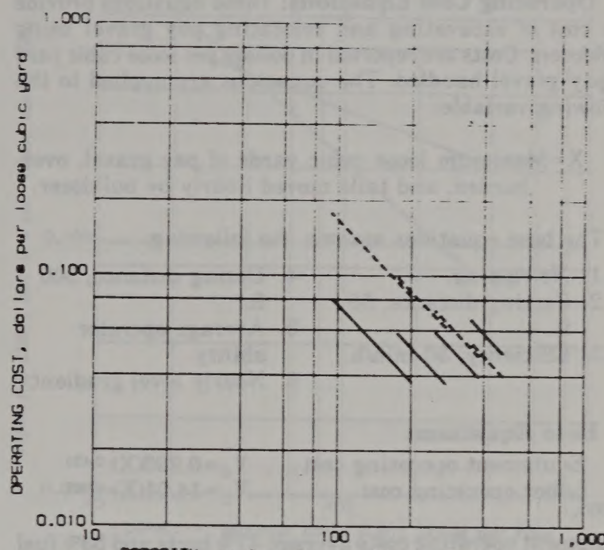
Digging Depth Factor: If average digging depth is other than 50% of maximum, the factor obtained from the following equation must be applied to the total cost per loose cubic yard of pay gravel:

$F_D=0.09194(\text{percent of maximum digging depth})^{0.608}$.

Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by the factors obtained from the following equations:

Equipment factor . . . . $U_e=1.078(X)^{-0.003}$
Labor factor . . . . $U_l=0.918(X)^{0.021}$

Digging Difficulty Factor: Parameters given in the discussion on site adjustment factors in section 1 should be used to determine if a digging difficulty factor is required.



Mining operating costs - Backhoes

If so, one of the following should be applied to total cost per loose cubic yard of pay gravel:

F_H, easy digging	1.000	F_H, medium-hard digging	1.500
F_H, medium digging	1.250	F_H, hard digging	1.886

Total Cost: Cost per loose cubic yard of pay gravel is determined by

$$[Y_e(U_e) + Y_L(U_l)] \times F_D \times F_H.$$

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *pay gravel* handled by backhoe. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.

MINING—BULLDOZERS

Operating Cost Equations: These equations provide the cost of excavating and relocating pay gravel using bulldozers. Costs are reported in dollars per loose cubic yard of pay gravel handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by bulldozer.

The base equations assume the following:

1. No ripping.
2. Cutting distance, 50 ft.
3. Efficiency, 50 min/h.
4. Dozing distance, 300 ft.
5. Average operator ability.
6. Nearly level gradient.

Base Equations:

Equipment operating cost . . . $Y_E=0.993(X)^{-0.430}$

Labor operating cost . . . . $Y_L=14.01(X)^{-0.945}$

Equipment operating costs average 47% parts and 53% fuel and lubrication. Labor operating costs average 86% operator labor and 14% repair labor.

Distance Factor: If average dozing distance is other than 300 ft, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_D=0.00581(\text{distance})^{0.904}$.

Gradient Factor: If average gradient is other than level, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_G=1.041e^{[0.015(\text{percent gradient})]}$.

Ripping Factor: If ripping is required, total operating cost must be multiplied by the following factor. This will account for reduced productivity associated with ripping:

$F_R=1.595$.

Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by the factors obtained from the following equations:

Equipment factor . . . . $U_e=1.206(X)^{-0.013}$

Labor factor . . . . $U_l=0.967(X)^{0.015}$

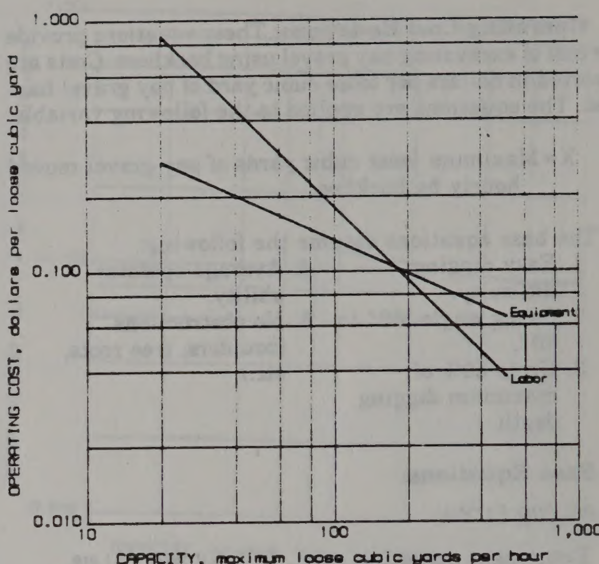
Digging Difficulty Factor: Parameters given in the discussion on site adjustment factors in section 1 should be used to determine if a digging difficulty factor is required If so, one of the following should be applied to total cost per loose cubic yard.

F_H, easy digging . . . 0.830

F_H, medium digging 1.000

F_H, medium-hard digging 1.250

F_H, hard digging 1.670



Mining operating costs - Bulldozers

Total Cost: Cost per loose cubic yard of pay gravel is determined by

$$[Y_E(U_e)+Y_L(U_l)]\times F_D\times F_G\times F_H\times F_R.$$

The total cost per loose cubic yard must then be multiplied by total yearly amount of *pay gravel* handled by bulldozer. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.

MINING—DRAGLINES

Operating Cost Equations: These equations provide the cost of excavating pay gravel using draglines. Costs are reported in dollars per loose cubic yard of pay gravel handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by dragline.

The base curves assume the following:

1. Bucket efficiency, 0.90.
2. Full hoist
3. Swing angle, 90°.
4. Average operator ability.

Base Equations:

Equipment operating cost . . . $Y_E = 1.984(X)^{-0.390}$
Labor operating cost . . . . $Y_L = 12.19(X)^{-0.888}$

Equipment operating costs consist of 67% parts and 33% fuel and lubrication. Labor operating costs consist of 78% operator labor and 22% repair labor.

Swing Angle Factor: If the average swing angle is other than 90°, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_S = 0.304(\text{swing angle})^{0.269}$.

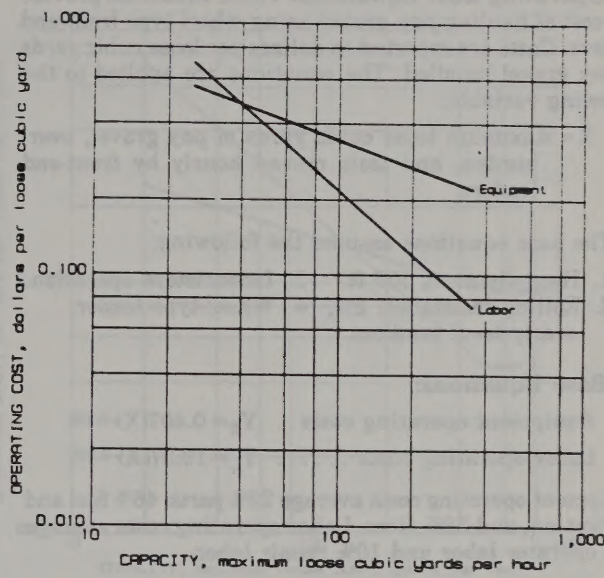
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by the factors obtained from the following equations:

Equipment factor . . . . $U_e = 1.162(X)^{-0.017}$
Labor factor . . . . $U_l = 0.989(X)^{0.006}$

Total Cost: Cost per loose cubic yard of pay gravel is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_S.$$

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *pay gravel* handled by dragline. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Mining operating costs - Draglines

MINING—FRONT-END LOADERS

Operating Cost Equations: These equations provide the cost of hauling pay gravel using wheel-type front-end loaders. Costs are reported in dollars per loose cubic yards of pay gravel handled. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by front-end loaders.

The base equations assume the following:

1. Haul distance, 500 ft.
2. Rolling resistance, 2%, nearly level gradient.
3. Inconsistent operation.
4. Wheel-type loader.

Base Equations:

Equipment operating costs . . $Y_E = 0.407(X)^{-0.225}$

Labor operating costs . . . . . . $Y_L = 13.07(X)^{-0.936}$

Equipment operating costs average 22% parts, 46% fuel and lubrication, and 32% tires. Labor operating costs average 90% operator labor and 10% repair labor.

Distance Factor: If the average haul distance is other than 500 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_D = 0.023(\text{distance})^{0.616}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_G = 0.877e^{[0.046(\text{percent gradient})]}$.

Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor . . . . $U_e = 1.162(X)^{-0.017}$
Labor factor . . . . $U_l = 0.989(X)^{0.006}$

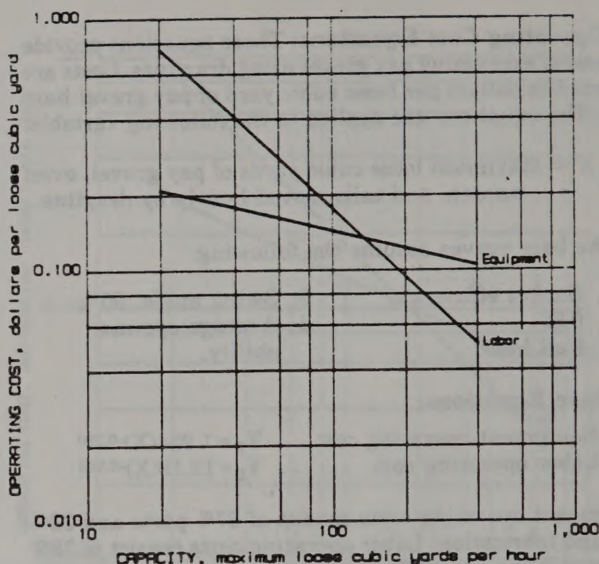
Track-Type Loader Factor: If track-type loaders are used, the following factors must be applied to total cost obtained from the base equations:

Equipment factor . . . . $T_e = 1.378$
Labor factor . . . . $T_l = 1.073$

Total Cost: Cost per loose cubic yard of pay gravel is determined by

$$[Y_E(U_e)(T_e) + Y_L(U_l)(T_l)] \times F_D \times F_G.$$

The total cost per loose cubic yard must then be multiplied by total yearly amount of *pay gravel* handled by front-end loader. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Mining operating costs - Front-end loaders

MINING—REAR-DUMP TRUCKS

Operating Cost Equations: These equations provide the cost of hauling pay gravel using rear-dump trucks. Costs are reported in dollars per loose cubic yard of pay gravel. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by rear dump truck.

The base equations assume the following:

1. Haul distance, 2,500 ft.
2. Loader cycles to fill, 4.
3. Efficiency, 50 min/h.
4. Average operator ability.
5. Rolling resistance, 2%, nearly level gradient.

Base Equations:

Equipment operating cost . . . $Y_E = 0.602(X)^{-0.296}$
Labor operating cost . . . . $Y_L = 11.34(X)^{-0.891}$

Equipment operating costs consist of 28% parts, 58% fuel and lubrication, and 14% tires. Labor operating costs consist of 82% operator labor and 18% repair labor.

Distance Factor: If average haul distance is other than 2,500 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_D = 0.093(\text{distance})^{0.311}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_G = 0.907e^{(0.049(\text{percent gradient}))}$.

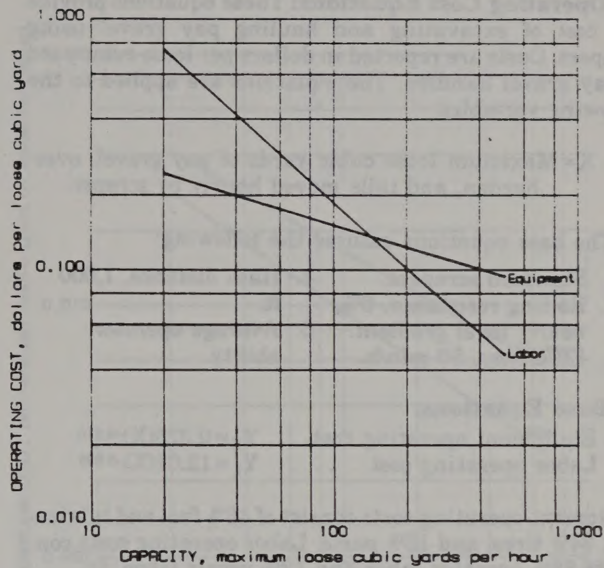
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor . . . . $U_e = 0.984(X)^{-0.016}$
Labor factor . . . . $U_l = 0.943(X)^{0.021}$

Total Cost: Cost per loose cubic yard of pay gravel is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_D \times F_G.$$

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *pay gravel* handled by rear-dump truck. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Mining operating costs - Rear-dump trucks

MINING—SCRAPERS

Operating Cost Equations: These equations provide the cost of excavating and hauling pay gravel using scrapers. Costs are reported in dollars per loose cubic yard of pay gravel handled. The equations are applied to the following variables:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by scraper.

The base equations assume the following:

1. Standard scrapers.
2. Rolling resistance, 6%, nearly level gradient.
3. Efficiency, 50 min/h.
4. Haul distance, 1,000 ft.
5. Average operator ability.

Base Equations:

Equipment operating cost . . . $Y_E=0.325(X)^{-0.210}$
Labor operating cost . . . . $Y_L=12.01(X)^{-0.930}$

Equipment operating costs consist of 48% fuel and lubrication, 34% tires, and 18% parts. Labor operating costs consist of 88% operator labor and 12% repair labor.

Distance Factor: If average haul distance is other than 1,000 ft, the factor obtained from the following equation must be applied to the total cost per loose cubic yard:

$F_D=0.01947(\text{distance})^{0.577}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 6%, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_G=0.776e^{[0.047(\text{percent gradient})]}$.

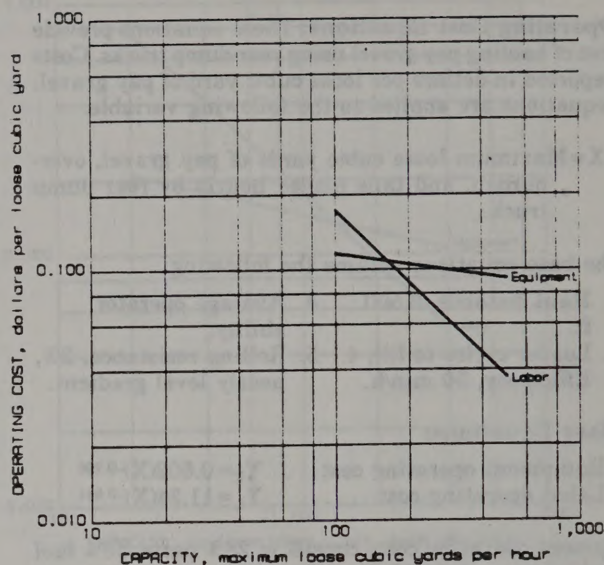
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor . . . . $U_e=1.096(X)^{-0.006}$
Labor factor . . . . $U_l=0.845(X)^{0.034}$

Total Cost: Cost per loose cubic yard of pay gravel is determined by

$$[Y_E(U_e)+Y_L(U_l)]\times F_D\times F_G.$$

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *pay gravel* handled by scraper. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Mining operating costs - Scrapers

PROCESSING—CONVEYORS

Operating Cost Equations: These equations provide the cost of moving gravel using conveyors. Costs are reported in dollars per cubic yard of gravel handled and include the operating cost of the conveyor along with the drive. The equations are applied to the following variable:

X = Maximum cubic yards of material moved hourly by conveyor.

The base equations assume the following:

1. Conveyors, 40 ft long.
2. Feed, 3,120 lb/yd³.
3. Nearly level setup.

Base Equations:

Equipment operating cost . . . $Y_E = 0.218(X)^{-0.561}$
Labor operating cost . . . . $Y_L = 0.250(X)^{-0.702}$

Equipment operating costs average 72% parts, 24% electricity, and 4% lubrication. Labor operating costs consist entirely of repair labor.

Conveyor Length Factor: If conveyor length is other than 40 ft, factors obtained from the following equations must be applied to respective portions of the operating costs. These factors are valid for conveyors 10 to 100 ft long:

Equipment factor . . . . $L_e = 0.209(\text{length})^{0.431}$
Labor factor . . . . $L_l = 0.245(\text{length})^{0.390}$

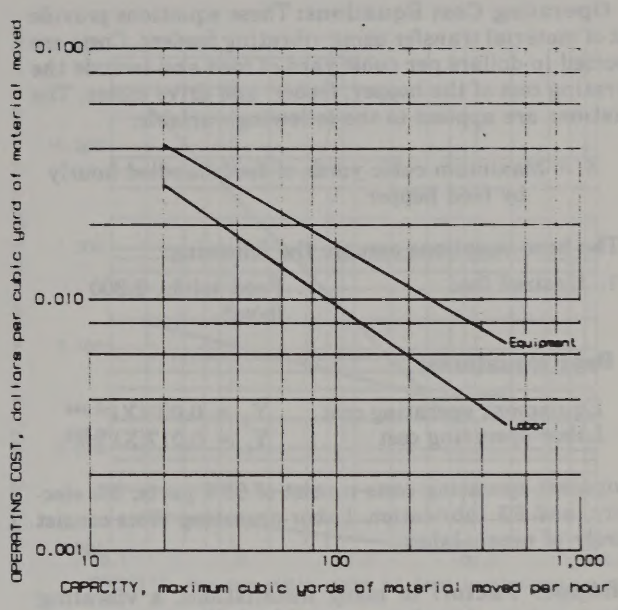
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of base operating costs must be multiplied by the following factors:

Equipment factor . . . . $U_e = 1.155$
Labor factor . . . . $U_l = 1.250$

Total Cost: Cost per cubic yard of gravel is determined by

$$[Y_E(L_e)(U_e) + Y_L(L_l)(U_l)].$$

The total cost per cubic yard must then be multiplied by the total yearly amount of *feed* handled by conveyor. (A separate operating and total yearly cost must be calculated for each conveyor in the circuit.) This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Conveyors

PROCESSING—FEED HOPPERS

Operating Cost Equations: These equations provide cost of material transfer using vibrating feeders. Costs are reported in dollars per cubic yard of feed and include the operating cost of the hopper, feeder, and drive motor. The equations are applied to the following variable:

X = Maximum cubic yards of feed handled hourly by feed hopper.

The base equations assume the following:

1. Unsized feed.
2. Feed solids, 2,300 lb/yd³.

Base Equations:

Equipment operating cost . . . $Y_E = 0.033(X)^{-0.344}$
Labor operating cost . . . . $Y_L = 0.017(X)^{-0.295}$

Equipment operating costs consist of 88% parts, 6% electricity, and 6% lubrication. Labor operating costs consist entirely of repair labor.

Hopper Factor: In many installations, a vibrating feeder is not used, and pay gravel feeds directly from the hopper. If this is the case, no operating cost for feeders is required.

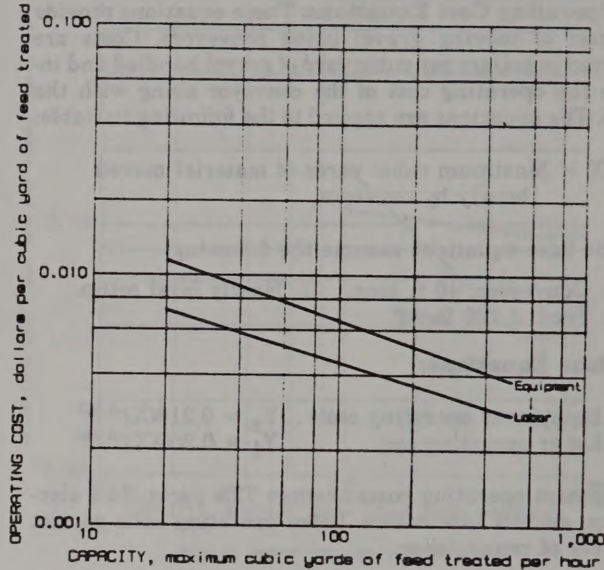
Used Equipment Factor: If a feeder with over 10,000 h of previous service life is to be used, the following factors must be applied to respective operating costs to account for increased maintenance and repair requirements:

Equipment factor . . . . $U_e = 1.176$
Labor factor . . . . $U_l = 1.233$

Total Cost: Cost per cubic yard of feed is determined by

$$[Y_E(U_e) + Y_L(U_l)].$$

The total cost per cubic yard must then be multiplied by total yearly amount of *feed* handled by feed hopper. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Feed hoppers

PROCESSING—JIG CONCENTRATORS

Operating Cost Equations: These equations provide the cost of gravity separation using jig concentrators. Costs are reported in dollars per cubic yard and include the operating cost of the jigs and associated drive motors. The equations are applied to the following variable:

X = Maximum cubic yards of feed handled hourly by jig concentrators.

The base equations assume the following:

1. Cleaner service.
2. Hourly capacity, 0.617 yd^3/ft^2.
3. Feed solids, 3,400 lb/yd^3.
4. Slurry density, 40% solids.
5. Gravity feed.

Base Equations:

Equipment operating cost . . . $Y_E = 0.113(X)^{-0.328}$
Supply operating cost . . . . . . $Y_S = 0.002(X)^{-0.184}$
Labor operating cost . . . . $Y_L = 3.508(X)^{-1.268}$

Equipment operating costs consist of 40% parts, 34% electricity, and 26% lubrication. Supply operating costs consist entirely of lead shot for bedding material. Labor operating costs consist of 66% operator labor and 34% repair labor.

Rougher-Coarse Factor: If jigs are to be used for rougher service or a coarse feed, higher productivity will be realized. To compensate for this situation, the following factor must be applied to total operating cost:

$F_R = 0.344.$

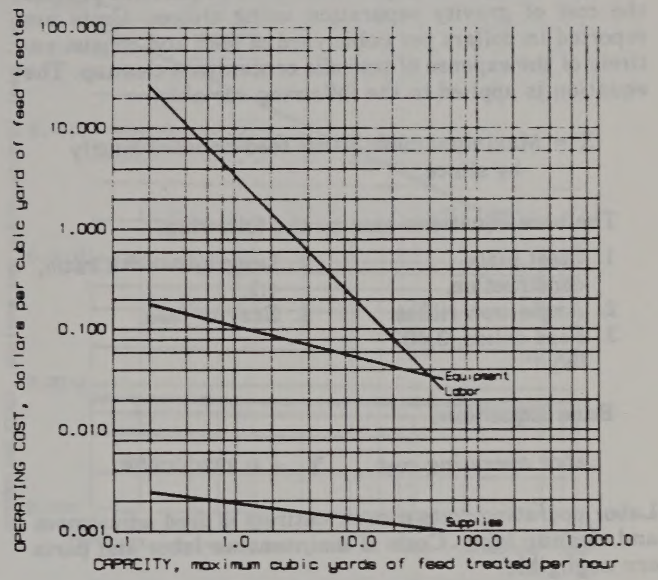
Used Equipment Factor: If jig concentrators with over 10,000 h of service life are to be used, the following factors must be applied to respective operating costs to account for increased maintenance and repair requirements:

Equipment factor . . . . $U_e = 1.096$
Labor factor . . . . $U_l = 1.087$

Total Cost: Cost per cubic yard of feed is determined by

$$[Y_E(U_e) + Y_S + Y_L(U_l)] \times F_R.$$

The total cost per cubic yard must then be multiplied by the total yearly amount of *feed* handled by jig concentrators. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Jig concentrators

PROCESSING—SLUICES

Operating Cost Equations: These equations provide the cost of gravity separation using sluices. Costs are reported in dollars per cubic yard of feed and consist entirely of the expense of periodic concentrate cleanup. The equation is applied to the following variable:

X = Maximum cubic yards feed handled hourly by sluice.

The base equations assume the following:

1. Steel plate construction.
2. Angle iron riffles.
3. Feed solids, 3,400 lb/yd³.
4. Length-to-width ratio, 4:1.
5. Gravity feed.

Base Equation:

Labor operating cost . . . $Y_L = 0.337(X)^{-0.636}$

Labor operating costs consist entirely of feed adjustment and cleanup labor. Costs of maintenance labor and parts are negligible.

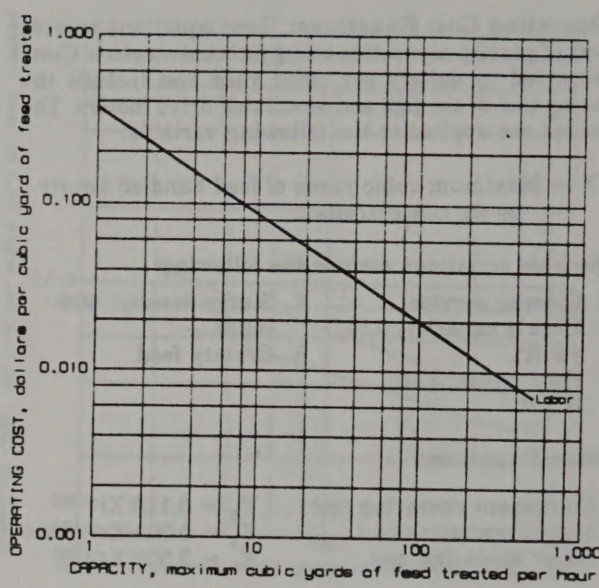
Wood Sluice Factor: If wood sluices are to be used, an allowance must be made for periodic sluice replacement. To account for this, an equipment cost must be added to total cost, and labor cost must be multiplied by the following factor:

Equipment cost . . . . $Y_E = 0.00035(X)^{0.383}$
Labor factor . . . . $W_l = 1.141$

Total Cost: Cost per cubic yard of feed is determined by

$$[Y_L(W_l) + Y_E].$$

The total cost per cubic yard must then be multiplied by total yearly amount of *feed* handled by sluices. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Sluices

PROCESSING—SPIRAL CONCENTRATORS

Operating Cost Equations: These equations provide the cost of gravity separation using spiral concentrators. Costs are reported in dollars per cubic yard of feed and include the operating cost of the spirals and slurry splitters only. The equations are applied to the following variable:

X = Maximum cubic yards of feed handled hourly by spiral concentrators.

The base equations assume the following:

1. Rougher service.
2. Solids per start, 1.75 st/h.
3. Feed solids, 3,400 lb/yd³.
4. Slurry density, 10% solids.
5. Gravity feed.

Base Equations:

Equipment operating cost... Y_E = \$0.0007/yd³
Labor operating cost .... $Y_L = 0.755(X)^{-0.614}$

Equipment operating costs consist entirely of parts. Labor operating costs consist entirely of operator labor, with the operator performing functions such as lining replacement.

Cleaner-Scavenger Factor: If spirals are to be used for cleaning or scavenging, throughput is reduced. The following factors must be applied to respective operating costs:

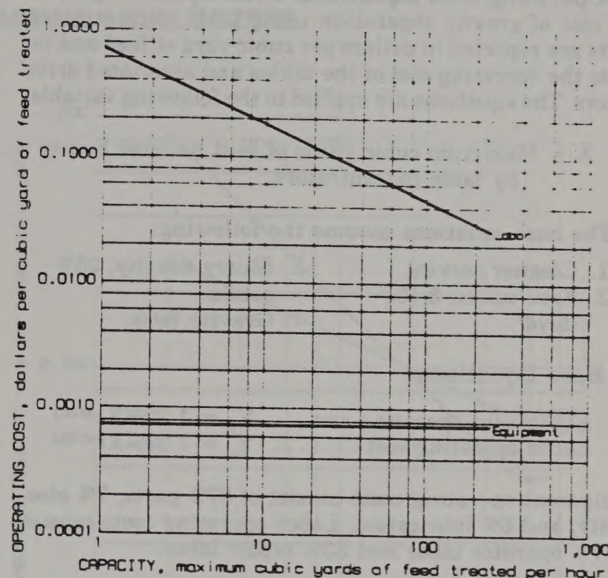
Equipment factor.......... $C_e = 2.429$
Labor factor .... $C_l = 1.796$

Used Equipment Factor: Because spiral concentrators have no moving parts, they enjoy a long service life. Generally, only the liners require periodic replacement. For this reason, the operating costs asscociated with spirals are typically constant throughout the life of the machine.

Total Cost: Cost per cubic yard of feed is determined by

$$[0.0007(C_e) + Y_L(C_l)].$$

The total cost per cubic yard must then be multiplied by the total yearly amount of *feed* handled by spiral concentrators. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Spiral concentrators

PROCESSING—TABLE CONCENTRATORS

Operating Cost Equations: These equations provide the cost of gravity separation using table concentrators. Costs are reported in dollars per cubic yard of feed and include the operating cost of the tables and associated drive motors. The equations are applied to the following variable:

X = Maximum cubic yards of feed handled hourly by table concentrators.

The base equations assume the following:

1. Cleaner service.
2. Feed solids, 3,400 lb/yd³.
3. Slurry density, 25% solids.
4. Gravity feed.

Base Equations:

Equipment operating cost . . . $Y_E = 1.326(X)^{-0.443}$
Labor operating cost . . . . $Y_L = 1.399(X)^{-0.783}$

Equipment operating costs consist of 87% parts, 7% electricity, and 6% lubrication. Labor operating costs consist of 67% operator labor and 33% repair labor.

Rougher-Coarse Factor: If the tables are to be used for rougher service or a coarse feed, higher productivity will be realized. To compensate for this situation, the following factors must be applied to both equipment and labor operating costs:

Equipment factor . . . . $R_e = 0.415$
Labor factor . . . . $R_l = 0.415$

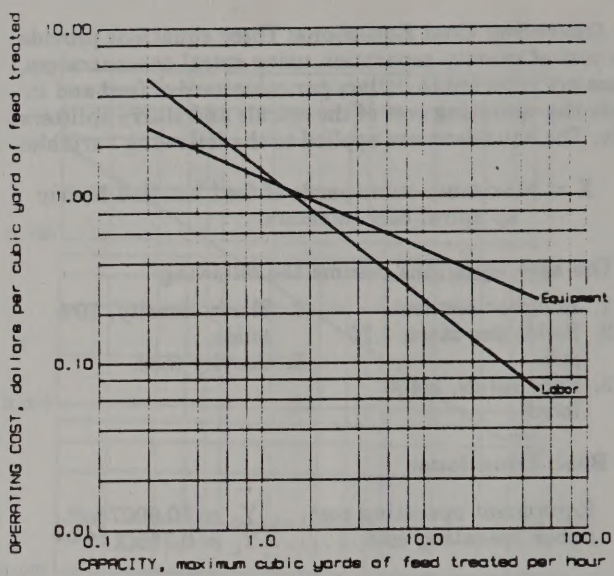
Used Equipment Factor: If table concentrators with over 10,000 h of service life are to be used, the following factors must be applied to the respective operating costs to account for increased maintenance and repair requirements:

Equipment factor . . . . $U_e = 1.217(X)^{-0.002}$
Labor factor . . . . $U_l = 1.121(X)^{-0.026}$

Total Cost: Cost per cubic yard of feed is determined by

$$[Y_E(R_e)(U_e)+Y_L(R_l)(U_l)].$$

The total cost per cubic yard must then be multiplied by the total yearly amount of *feed* handled by table concentrators. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Table concentrators

PROCESSING—TAILINGS REMOVAL—BULLDOZERS

Operating Cost Equations: These equations provide the cost of removing and relocating tailings using bulldozers. Costs are reported in dollars per cubic yard of tailings moved. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by bulldozer.

The base equations assume the following:

1. Efficiency, 50 min/h.
2. Dozing distance, 300 ft.
3. Average operator ability.
4. Nearly level gradient.

Base Equations:

Equipment operating cost . . . $Y_E = 0.993(X)^{-0.430}$

Labor operating cost . . . . $Y_L = 14.01(X)^{-0.945}$

Equipment operating costs average 47% parts, and 53% fuel and lubrication. Labor operating costs average 86% operator labor and 14% repair labor.

Distance Factor: If average dozing distance is other than 300 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_D = 0.00581(\text{distance})^{0.904}$.

Gradient Factor: If average gradient is other than level, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_G = 1.041e^{[0.015(\text{percent gradient})]}$.

Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

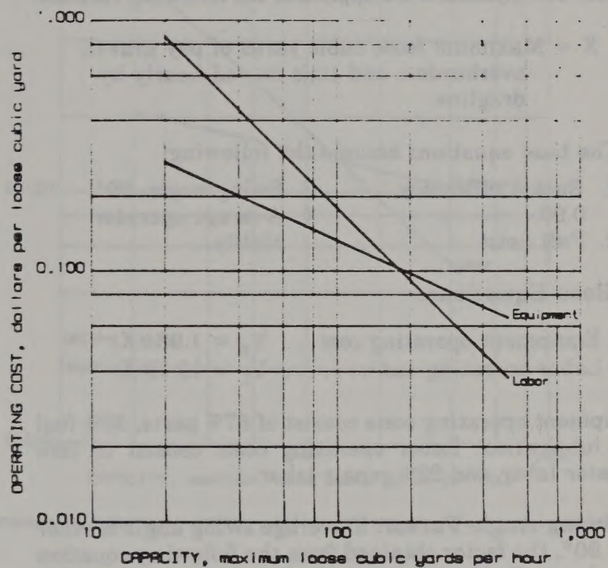
Equipment factor . . . . $U_e = 1.206(X)^{-0.013}$

Labor factor . . . . $U_l = 0.967(X)^{0.015}$

Total Cost: Cost per cubic yard of tailings is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_D \times F_G.$$

The total cost per cubic yard must then be multiplied by the total yearly amount of *tailings* moved by bulldozer. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Tailings removal - Bulldozers

PROCESSING—TAILINGS REMOVAL—DRAGLINES

Operating Cost Equations: These equations provide the cost of removing and relocating tailings using draglines. Costs are reported in dollars per cubic yard of tailings moved. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by dragline.

The base equations assume the following:

1. Bucket efficiency, 0.90.
2. Full hoist.
3. Swing angle, 90°.
4. Average operator ability.

Base Equations:

Equipment operating cost . . . $Y_E = 1.984(X)^{-0.390}$
Labor operating cost . . . . $Y_L = 12.19(X)^{-0.888}$

Equipment operating costs consist of 67% parts, 33% fuel and lubrication. Labor operating costs consist of 78% operator labor and 22% repair labor.

Swing Angle Factor: If average swing angle is other than 90°, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_S = 0.304(\text{swing angle})^{0.269}$.

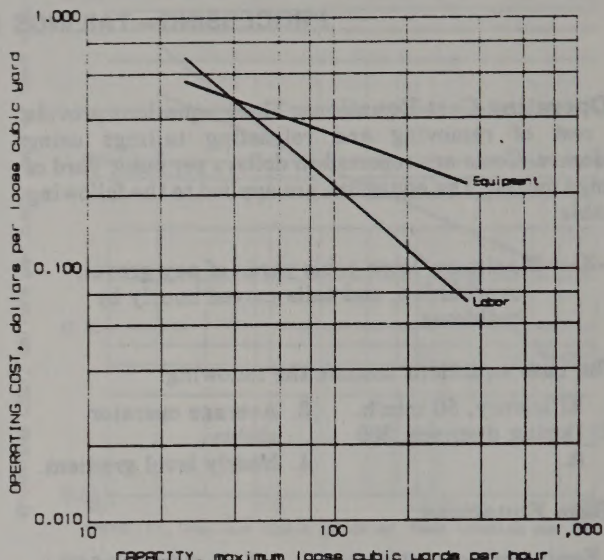
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor . . . . $U_e = 1.162(X)^{-0.017}$
Labor factor . . . . $U_l = 0.989(X)^{0.006}$

Total Cost: Cost per cubic yard of feed is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_S.$$

The total cost per cubic yard must then be multiplied by the total yearly amount of *tailings* moved by dragline. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Tailings removal - Draglines

PROCESSING—TAILINGS REMOVAL—FRONT-END LOADERS

Operating Cost Equations: These equations provide the cost of removing and relocating tailings using wheel-type front-end loaders. Costs are reported in dollars per cubic yard of tailings moved. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by front-end loader.

The base equations assume the following:

1. Haul distance, 500 ft.
2. Rolling resistance, 2%, nearly level gradient.
3. Inconsistent operation.
4. Wheel-type loader.

Base Equations:

Equipment operating cost... $Y_E = 0.407(X)^{-0.225}$
Labor operating cost .... $Y_L = 13.07(X)^{-0.936}$

Equipment operating costs average 22% parts, 46% fuel and lubrication, and 32% tires. Labor operating costs average 90% operator labor and 10% repair labor.

Distance Factor: If average haul distance is other than 500 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_D = 0.023(\text{distance})^{0.616}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_G = 0.877e^{[0.046(\text{percent gradient})]}$.

Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor......... $U_e = 1.162(X)^{-0.017}$
Labor factor .... $U_l = 0.989(X)^{0.006}$

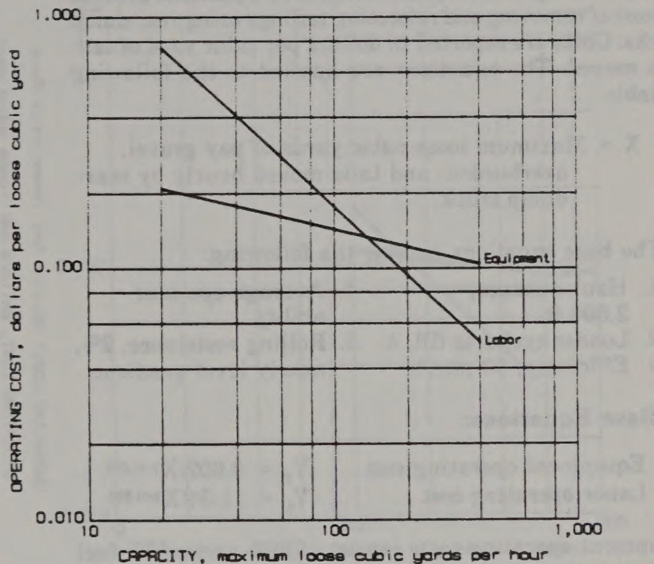
Track-Type Loader Factor: If track-type loaders are used, the following factors must be applied to total cost obtained from the base equations:

Equipment factor .... $T_e = 1.378$
Labor factor .... $T_l = 1.073$

Total Cost: Cost per cubic yard of tailings is determined by

$$[Y_E(U_e)(T_e) + Y_L(U_l)(T_l)] \times F_D \times F_G.$$

The total cost per cubic yard must then be multiplied by the total yearly amount of *tailings* moved by front-end loader. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - tailings removal - Front-end loaders

PROCESSING—TAILINGS REMOVAL—REAR-DUMP TRUCKS

Operating Cost Equations: These equations provide the cost of removing and relocating tailings using rear-dump trucks. Costs are reported in dollars per cubic yard of tailings moved. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by rear-dump truck.

The base equations assume the following:

1. Haul distance, 2,500 ft.
2. Loader cycles to fill, 4.
3. Efficiency, 50 min/h.
4. Average operator ability.
5. Rolling resistance, 2%, nearly level gradient.

Base Equations:

Equipment operating cost . . . $Y_E = 0.602(X)^{-0.296}$
Labor operating cost . . . . $Y_L = 11.34(X)^{-0.891}$

Equipment operating costs consist of 28% parts, 58% fuel and lubrication, and 14% tires. Labor operating costs consist of 82% operator labor and 18% repair labor.

Distance Factor: If average haul distance is other than 2,500 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_D = 0.093(\text{distance})^{0.311}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 2%, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_G = 0.907e^{(0.049 \text{ percent gradient})}$.

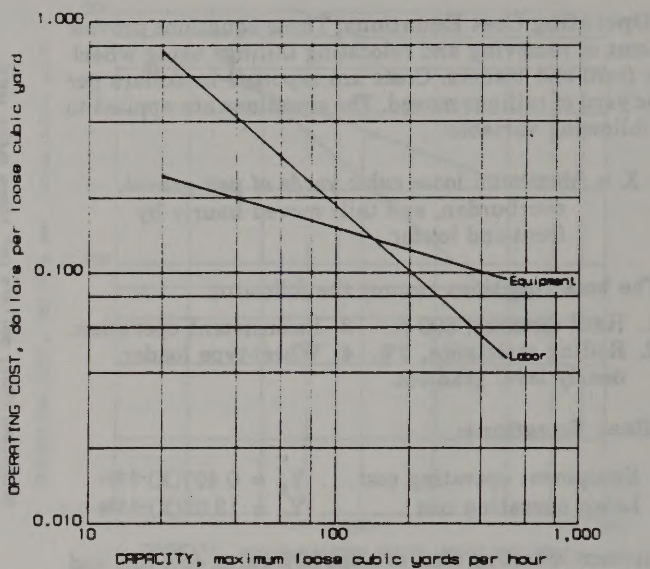
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of the base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor . . . . $U_e = 0.984(X)^{0.016}$
Labor factor . . . . $U_l = 0.943(X)^{0.021}$

Total Cost: Cost per cubic yard of tailings is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_D \times F_G.$$

The total cost per cubic yard must then be multiplied by the total yearly amount of *tailings* moved by truck. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Tailings removal - Rear-dump trucks

OPERATING COSTS

PROCESSING—TAILINGS REMOVAL—SCRAPERS

Operating Cost Equations: These equations provide the cost of removing and relocating tailings using scrapers. Costs are reported in dollars per cubic yard of tailings moved. The equations are applied to the following variable:

X = Maximum loose cubic yards of pay gravel, overburden, and tails moved hourly by scraper.

The base curves assume the following:

1. Standard scrapers.
2. Rolling resistance, 6%, nearly level gradient.
3. Efficiency, 50 min/h.
4. Haul distance, 1,000 ft.
5. Average operator ability.

Base Equation:

Equipment operating cost . . . $Y_E = 0.325(X)^{-0.210}$
Labor operating cost . . . . $Y_L = 12.01(X)^{-0.930}$

Equipment operating costs consist of 48% fuel and lubrication, 34% tires, and 18% parts. Labor operating costs consist of 88% operator labor and 12% repair labor.

Distance Factor: If average haul distance is other than 1,000 ft, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_D = 0.01947(\text{distance})^{0.577}$.

Gradient Factor: If total gradient (gradient plus rolling resistance) is other than 6%, the factor obtained from the following equation must be applied to total cost per loose cubic yard:

$F_G = 0.776e^{[0.047(\text{percent gradient})]}$.

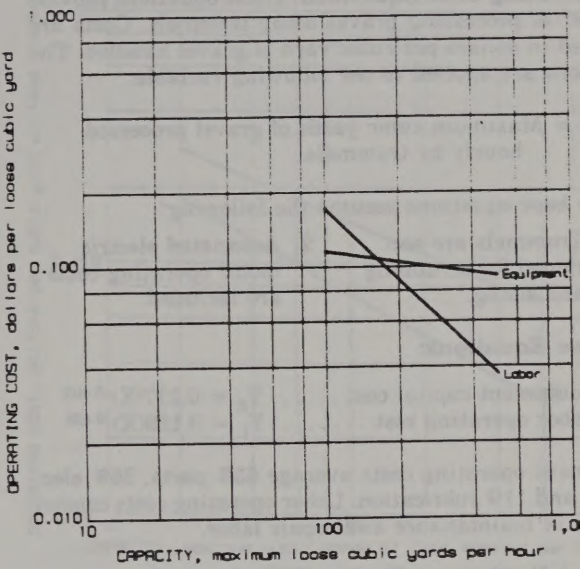
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of base operating costs must be multiplied by factors obtained from the following equations:

Equipment factor . . . . $U_e = 1.096(X)^{-0.006}$
Labor factor . . . . $U_l = 0.845(X)^{0.034}$

Total Cost: Cost per cubic yard of tailings is determined by

$$[Y_E(U_e) + Y_L(U_l)] \times F_D \times F_G.$$

The total cost per cubic yard must then be multiplied by total yearly amount of *tailings* moved by scraper. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Tailings removal - Scrapers

PROCESSING—TROMMELS

Operating Cost Equations: These equations provide the cost of processing gravel using trommels. Costs are reported in dollars per cubic yard of gravel handled. The equations are applied to the following variable:

X = Maximum cubic yards of gravel processed hourly by trommels.

The base equations assume the following:

1. Trommels are sectioned for scrubbing and sizing.
2. Associated electric motor operating costs are included.

Base Equations:

Equipment capital cost ..... $Y_E = 0.217(X)^{-0.403}$
Labor operating cost .... $Y_L = 0.129(X)^{-0.429}$

Equipment operating costs average 63% parts, 26% electricity, and 11% lubrication. Labor operating costs consist entirely of maintenance and repair labor.

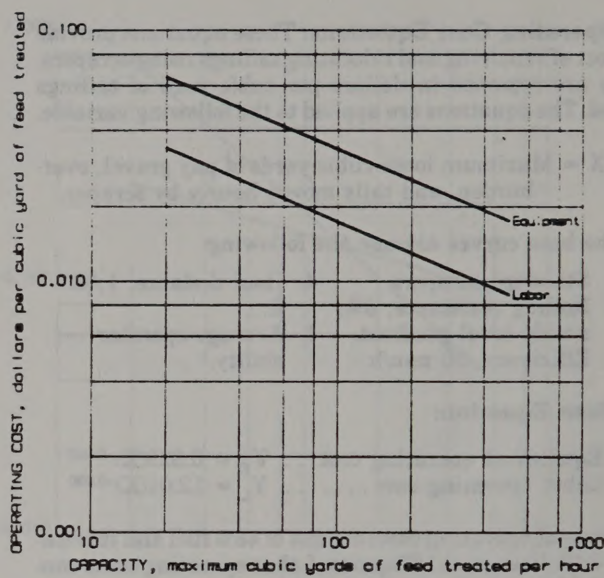
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of base operating costs must be multiplied by the following factors:

Equipment factor .... $U_e = 1.194$
Labor factor .... $U_l = 1.310$

Total Cost: Cost per cubic yard of gravel is determined by

$$[Y_E(U_e) + Y_L(U_l)].$$

The total cost per cubic yard must then be multiplied by the total yearly amount of gravel processed by trommels. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Trommels

PROCESSING—VIBRATING SCREENS

Operating Cost Equations: These equations provide the cost of processing gravel using vibrating screens. Costs are reported in dollars per cubic yard of gravel handled. The equations are applied to the following variable:

X = Maximum cubic yards of gravel processed hourly by vibrating screen.

The base equations assume the following:

1. An average of 0.624 ft^2 of screen is required for every cubic yard of hourly capacity.
2. Associated electric motor operating costs are included.
3. Feed solids, 3,120 lb/yd^3.
4. Gravity feed.

Base Equations:

Equipment operating cost . . . $Y_E = 0.104(X)^{-0.426}$
Labor operating cost . . . . $Y_L = 0.106(X)^{-0.570}$

Equipment operating costs average 73% parts, 19% electricity, and 8% lubrication. Labor operating costs consist entirely of maintenance and repair labor.

Capacity Factor: If anticipated screen capacity is other than 0.624 ft^2/yd^3 hourly feed capacity, the respective operating costs must be multiplied by factors obtained from the following equations:

$$C_e = 1.267(C)^{0.575},$$

and

$$C_l = 1.207(C)^{0.458},$$

where C = anticipated capacity in square feet of screen per cubic yard of hourly feed.

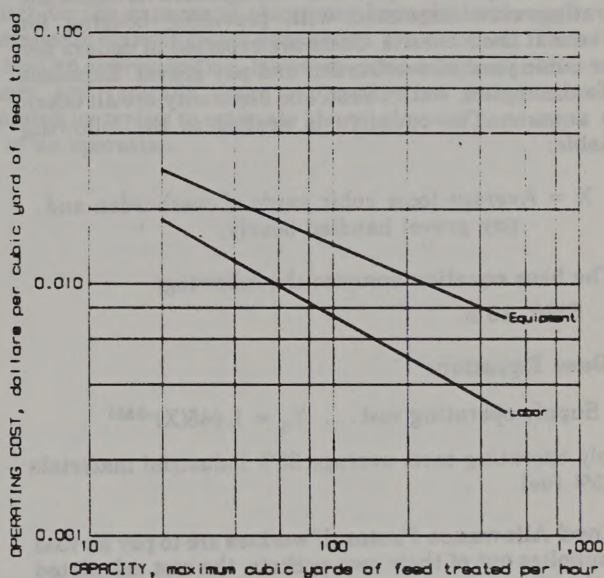
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of base operating costs must be multiplied by the following factors:

Equipment factor . . . . $U_e = 1.197$
Labor factor . . . . $U_l = 1.131$

Total Cost: Cost per cubic yard of gravel is determined by

$$[Y_E(C_e)(U_e) + Y_L(C_l)(U_l)].$$

The total cost per cubic yard must then be multiplied by the total yearly amount of gravel processed by the vibrating screen. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Processing operating costs - Vibrating screens

SUPPLEMENTAL—EMPLOYEE HOUSING

Operating Cost Equation: This equation furnishes the operating cost associated with providing housing for workers at the minesite. Costs are reported in dollars per loose cubic yard of overburden and pay gravel. Expenses for food, supplies, water, heat, and electricity are all taken into account. The equation is applied to the following variable:

X = Average loose cubic yards of overburden and pay gravel handled hourly.

The base equation assumes the following:

1. Shift, 10 h.

Base Equation:

Supply operating cost . . . $Y_S = 1.445(X)^{-0.583}$

Supply operating costs average 95% industrial materials and 5% fuel.

Food Allowance Factor: If workers are to pay for food and supplies out of their own pockets, the cost calculated from the above equation must be multiplied by the following factor:

$F_F = 0.048.$

Workforce Factor: The equation used to compute labor for operating cost estimation is

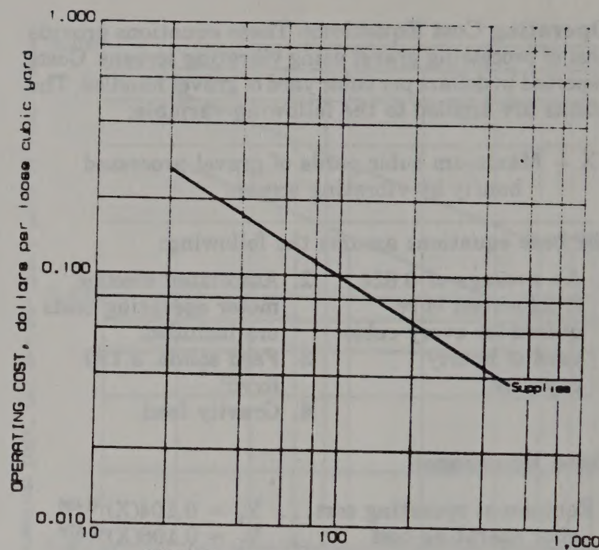
Workforce $= 0.822(X)^{0.415}$.

If the workforce for the operation under evaluation is known, and is different than that calculated from the above equation, the correct cost can be obtained from the following equation:

$$Y_S = \frac{(\text{Number of workers}) \times \$17.85}{\text{Cubic yards of overburden and pay gravel handled daily}}.$$

Total Cost: Cost per loose cubic yard is determined by $Y_S \times F_F$.

The total cost per loose cubic yard must then be multiplied by the total yearly amount of *overburden* and *pay gravel* handled. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Supplemental operating costs - Employee housing

SUPPLEMENTAL—GENERATORS

Operating Cost: Operating costs of diesel generators are accounted for in the electrical portions of the other equipment operating costs. By so doing, operating costs of the generators are tied directly to size and type of equipment used.

The electrical portions of operating cost curves will also account for the expense of electricity brought in through transmission lines if diesel generators are not used. This is at best an approximation. However, costs assigned in this manner are typically more representative than costs calculated by trying to estimate the total power consumption of an operation.

SUPPLEMENTAL—LOST TIME AND GENERAL SERVICES

Operating Cost Equations: These equations account for costs not directly related to production. Costs are reported in dollars per cubic yard. Items in this section include:

1. Equipment downtime.
   a. Productivity lost by the entire crew due to breakdown of key pieces of equipment.
   b. Productivity lost by individual operators due to breakdown of single pieces of equipment.
   c. Labor charges of outside maintenance personnel.
   d. Wash plant relocation.
2. Site maintenance.
   a. Road maintenance.
   b. Stream diversion.
   c. Drainage ditch construction and maintenance.
   d. Site cleanup.
   e. Reclamation grading and recontouring.
   f. Settling pond maintenance.
3. Concentrate refinement.
   a. Time spent recovering valuable minerals from mill concentrates by panning, mechanical separation, or amalgamation.

The equations are applied to the following variable:

X = Maximum cubic yards of feed handled hourly by mill.

Base Equations:

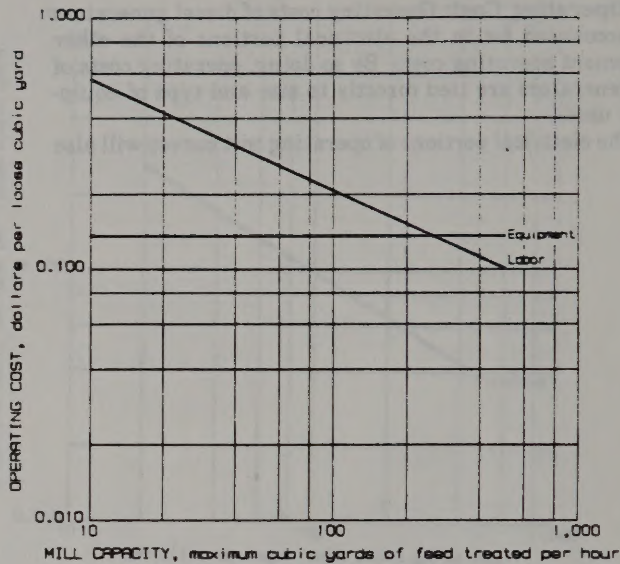
Equipment operating cost . . . $Y_E = 0.142(X)^{0.004}$
Labor operating cost . . . . $Y_L = 2.673(X)^{-0.524}$

Equipment operating costs average 53% fuel and lubrication and 47% equipment parts. Labor operating costs consist of 91% operator labor and 9% maintenance and repair labor.

Total Cost: Cost per cubic yard is determined by

$$Y_E + Y_L$$

The total cost per cubic yard must then be multiplied by the total yearly amount of *overburden, pay gravel,* and *tailings* handled. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Supplemental operating costs - Lost time and general services

SUPPLEMENTAL—PUMPS

Operating Cost Equations: These equations provide the cost of transporting and providing water using centrifugal pumps. Costs are reported in dollars per hour of pump use. If more than one pump is used in the operation, a separate cost must be calculated for each. The equations are applied to the following variable:

X = Maximum gallons of water required per minute.

The base equations assume the following:

1. Total head, 25 ft.
2. Diesel-powered pumps.
3. Abrasion-resistant steel construction.
4. Total engine-pump efficiency, 60%.

Base Equations:

Equipment operating cost . . . $Y_E = 0.007(X)^{0.713}$
Labor operating cost . . . . $Y_L = 0.004(X)^{0.867}$

Equipment operating costs average 59% fuel and lubrication, and 41% parts. Labor operating costs consist of 82% operator labor and 18% maintenance and repair labor. (Operator labor includes pipeline work.)

Head Factor: If total pumping head is other than 25 ft, factors calculated from the following equations will correct for changes in equipment and labor operating costs. The product of these factors and the original costs will provide the appropriate figures:

$$H_e = 0.091(H)^{0.735},$$

and

$$H_l = 0.054(H)^{0.893}$$

where H = total pumping head.

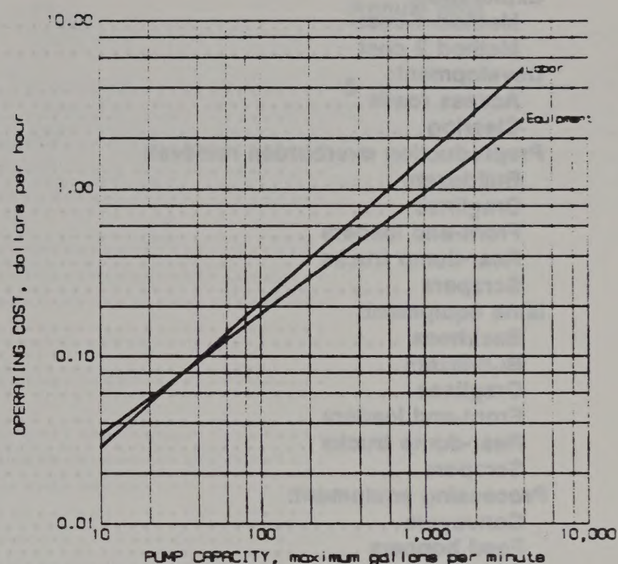
Used Equipment Factor: These factors account for added operating expenses accrued by equipment having over 10,000 h of previous service life. The respective equipment and labor portions of base operating costs must be multiplied by the following factors:

Equipment factor . . . . $U_e = 1.096$
Labor factor . . . . $U_l = 1.067$

Total Cost: Cost per hour is determined by

$$[Y_E(H_e)(U_e)] + [Y_L(H_l)(U_l)].$$

The total cost per hour must then be multiplied by the anticipated hours per year of pump use. This product is subsequently entered in the appropriate row of the tabulation shown in figure 6 for final operating cost calculation.



Supplemental operating costs - Pumps

CAPITAL COST SUMMARY FORM

Item	Cost
Exploration:	
Method 1 cost	$
Method 2 cost	
Development:	
Access roads	
Clearing	
Preproduction overburden removal:	
Bulldozers	
Draglines	
Front-end loaders	
Rear-dump trucks	
Scrapers	
Mine equipment:	
Backhoes	
Bulldozers	
Draglines	
Front-end loaders	
Rear-dump trucks	
Scrapers	
Processing equipment:	
Conveyors	
Feed hoppers	
Jig concentrators	
Sluices	
Spiral concentrators	
Table concentrators	
Trommels	
Vibrating screens	
Supplemental:	
Buildings	
Camp	
Generators	
Pumps	
Settling ponds	
Subtotal	
Contingency (10%)	
Total	

Figure 5.—Capital cost summary form.

OPERATING COST SUMMARY FORM

Item	Annual cost
Overburden removal:	
Bulldozers	$
Draglines	
Front-end loaders	
Rear-dump trucks	
Scrapers	
Mining:	
Backhoes	
Bulldozers	
Draglines	
Front-end loaders	
Rear-dump trucks	
Scrapers	
Processing:	
Conveyors	
Feed hoppers	
Jig concentrators	
Sluices	
Spiral concentrators	
Table concentrators	
Tailings removal:	
Bulldozers	
Draglines	
Front-end loaders	
Rear-dump trucks	
Scrapers	
Trommels	
Vibrating screens	
Supplemental:	
Employee housing	
Lost time and general services	
Pumps	
Subtotal	
Contingency (10%)	
Total	

Cost per cubic yard pay gravel = total annual cost divided by pay gravel mined per year.

Final cost per cubic yard pay gravel ....

Figure 6.—Operating cost summary form.

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APPENDIX.—EXAMPLE OF COST ESTIMATE

SAMPLE ESTIMATION

Parameters

General:
- 150 operating d/a.
- 10 h/shift.
- 100-LCY/h pay gravel capacity.
- 2.5-LCY stripping ratio.
- 150,000 LCY/a pay gravel mined.
- 375,000 LCY/a overburden removed.
- Workers live on site.

Exploration:
- 20 worker-days reconnaissance.
- 1,400 ft churn drilling.
- 2,000 yd^3 trenching
- 1,200 samples panned.
- 8 h helicopter time.
- 180 worker-days camp requirements.

Development:
- 4-mile access road.
- 20% side slope.
- Forested.
- 22 ft wide.
- Ungraveled.
- 6 acres cleared.
- Forested.
- 10% side slope.

Overburden removal:
- Excavated and hauled by 1 scraper.
- 250-LCY/h production capacity.
- 330,000 LCY prior to production.
- 3,000-ft average haul distance.
- +8% average haul gradient.

Mining:
- Excavation by 1 backhoe.
- Hauled by 2 front-end loaders.
- 100-LCY/h production capacity.
- Medium-hard digging.
- 800-ft average haul length.
- +6% average haul gradient.

Mine equipment:
- 1 new backhoe.
- 1 used bulldozer.
- 2 new front-end loaders.
- 1 used scraper.

Milling (jig plant, see figure A-1):
- Feeder, 100 LCY/h.
- Trommel, 100 LCY/h.
- Rougher jig, 20 yd^3/h.
- Cleaner jigs, 2 at 5 yd^3/h.
- Final jig, 0.2 yd^3/h.
- Scavenger sluice, 50 yd^3/h.
- Scavenger sluice, 20 yd^3/h.
- Conveyor, 70 yd^3/h, 40 ft.

Tailings placement:
- Transported using 1 bulldozer.
- 100-LCY/h production capacity.
- 400-ft average haul length.
- −8% average gradient.

Escalation Factors (January to July 1985)

Labor	11.98/11.69	= 1.025
Equipment	362.3/360.4	= 1.005
Steel	354.6/357.4	= 0.992
Lumber	354.9/343.2	= 1.034
Fuel	630.7/636.2	= 0.991
Tires	246.0/262.0	= 0.939
Construction materials	363.63/358.32	= 1.015
Electricity	540.3/524.9	= 1.029
Industrial materials	324.3/323.2	= 1.003

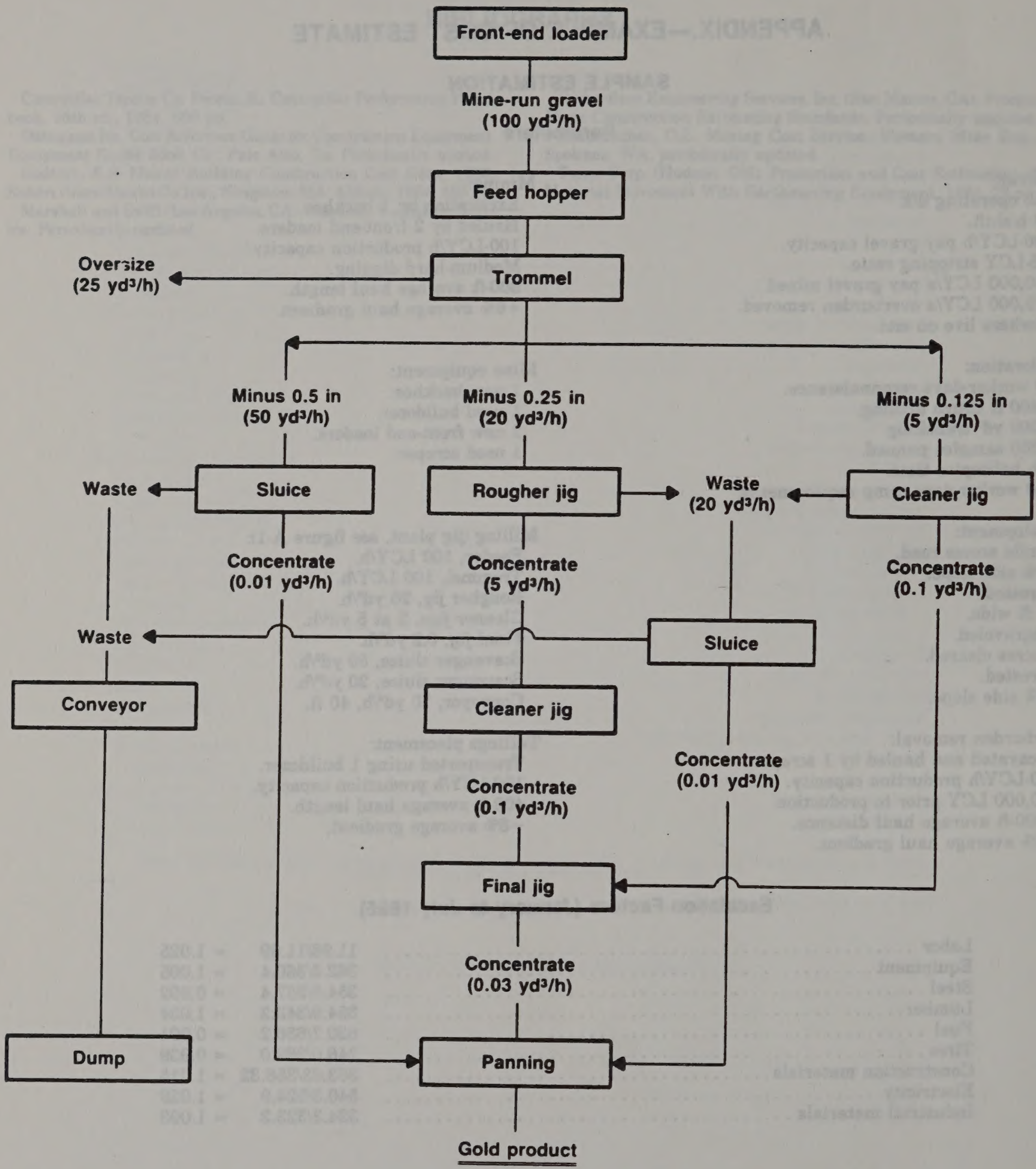

Figure A-1—Sample flow sheet.

CAPITAL COSTS

Exploration (p. 20)

Reconnaissance	20 worker-days × $195/worker-day	=	$3,900
Churn drilling	1,400 ft × $45/ft	=	63,000
Trenching	2,000 yd³ × $7.10/yd³	=	14,200
Panning	1,200 samples × $2.10/sample	=	2,520
Helicopter	8 h × $395/h	=	3,160
Camp	180 worker-days × $30/worker-day	=	5,400

Exploration capital cost = $92,180 × 1.025 (labor) $94,485

Access roads (p. 22)

22-ft wide
4 miles long
20% side slope
Forested
600 ft blasting

Base cost	$Y_C = 765.65(22)^{0.922}$	=	$13,236/mile
Labor	$13,236 × 0.68 × 1.025	=	$9,225
Parts	$13,236 × 0.13 × 1.005	=	1,729
Fuel	$13,236 × 0.16 × 0.991	=	2,099
Tires	$13,236 × 0.03 × 0.939	=	373
			$13,426/mile
Forest factor	$F_F = 2.000(22)^{-0.079}$	=	1.567
Side slope factor	$F_S = 0.633e^{[0.021(20)]}$	=	0.963
Blasting factor	$F_H = [12{,}059.18(22)^{0.534}] \times (600/5{,}280)$	=	7,140

Access road capital cost = [($13,426 × 1.567 × 0.963 × 4] + 7,140 .... $88,180

Clearing (p. 23)

6 acres
10% side slope
Forested

Base cost	$Y_C = 1{,}043.61(6)^{0.913}$	=	$5,358
Labor	$5,358 × 0.68 × 1.025	=	$3,735
Parts	$5,358 × 0.12 × 0.991	=	637
Fuel	$5,358 × 0.18 × 1.006	=	970
Steel	$5,358 × 0.02 × 0.992	=	106
			$5,448
Slope factor	$F_S = 0.942e^{[0.008(10)]}$	=	1.020
Forest factor	F_F	=	1.750

Clearing capital cost = [$5,448 × 1.020 × 1.750] $9,725

Preproduction overburden removal (p. 24)

30,000 LCY
250 LCY/h
3,000-ft haul
+8% haul gradient plus rolling resistance
Used scraper

Equipment cost	$Y_E = 0.325(250)^{-0.210}$	=	$0.102/LCY
Parts	$0.102 × 0.18 × 1.005	=	$0.018
Fuel and lubrication	$0.102 × 0.48 × 0.991	=	0.049
Tires	$0.102 × 0.34 × 0.939	=	0.033
			$0.100/LCY
Labor cost	$Y_L = 12.01(250)^{-0.930}$	=	$0.071/LCY
Labor	$0.071 × 1.00 × 1.025	=	$0.073/LCY
Distance factor	$F_D = 0.01947(3{,}000)^{0.577}$	=	1.975
Gradient factor	$F_G = 0.776e^{[0.047(8)]}$	=	1.130
Used equipment	$U_e = 1.096(250)^{-0.006}$	=	1.060
	$U_l = 0.845(250)^{0.034}$	=	1.019

Overburden removal capital cost = [($0.100 × 1.060) + ($0.073 × 1.019)] × 1.975 × 1.130 × 30,000 .... $12,077

Mine equipment—backhoes (p. 29)

100 LCY/h
80% maximum digging depth
Medium-hard digging

Base cost	$Y_C = 84{,}132.01e^{[0.00350(100)]}$	=	$119,389
Equipment	$119,389 × 1.00 × 1.005	=	$119,986
Digging depth factor	$F_D = 0.04484(80)^{0.790}$	=	1.429
Digging difficulty factor	F_H	=	1.556

Backhoe capital cost = ($119,986 × 1.429 × 1.556) $266,792

Mine equipment—bulldozers (p. 30)

100 LCY/h
400-ft average haul distance
−8% average haul gradient
Used equipment

Base cost	$Y_C = 3{,}555.96(100)^{0.806}$	=	$145,531
Equipment	$145,531.00 × 1.00 × 1.005	=	$146,259
Distance factor	$F_D = 0.01549(400)^{0.732}$	=	1.244
Gradient factor	$F_G = 1.041e^{[0.015(-8)]}$	=	0.923
Used equipment factor	F_U	=	0.411

Bulldozer capital cost = ($146,259 × 1.244 × 0.923 × 0.411) $69,022

Mine equipment—front-end loaders (p. 32)

100 LCY/h
Two machines, 50 yd³/h each
800-ft average haul
+6% haul gradient plus rolling resistance

Base cost . . . . $Y_C = 2{,}711.10(50)^{0.836}$ = $90,245

Equipment . . . . $90,245 × 1.00 × 1.005 = $90.696

Distance factor . . . . $F_D = 0.033(800)^{0.552}$ = 1.321
Gradient factor . . . . $F_G = 0.888e^{[0.041(6)]}$ = 1.136

Front-end loader capital cost = (2 × $90,696 × 1.321 × 1.136) $272,207

Mine equipment—scrapers (p. 34)

250 LCY/h
3,000-ft average haul
+8% haul gradient plus rolling resistance
Used equipment

Base cost . . . . $Y_C = 1{,}744.42(250)^{0.934}$ = $302,919

Equipment . . . . = $302,919 × 1.00 × 1.005 = $304,434

Distance factor . . . . $F_D = 0.025(3{,}000)^{0.539}$ = 1.871
Gradient factor . . . . $F_G = 0.776e^{[0.047(8)]}$ = 1.130
Used equipment factor . . . . F_U = 0.312

Scraper capital cost = ($304,434 × 1.871 × 1.130 × 0.312) $200,817

Processing equipment—conveyors (p. 35)

70 yd³/h
40 ft long

Base cost . . . . $Y_C = 4{,}728.36(70)^{0.287}$ = $16,005

Equipment price . . . . $16,005 × 0.89 × 1.005 = $14,316
Installation labor . . . . $16,005 × 0.08 × 1.025 = 1,312
Construction materials . . . . $16,005 × 0.03 × 1.015 = 487

Conveyor capital cost = ($14,316 + $1,312 + $487) $16,115

Processing equipment—feed hoppers (p. 36)

100 yd³/h

Base cost . . . . $Y_C = 458.48(100)^{0.470}$ = $3,993

Equipment price . . . . $3,993 × 0.82 × 1.005 = $3,291
Installation labor . . . . $3,993 × 0.14 × 1.025 = 573
Steel . . . . $3,993 × 0.04 × 0.992 = 158

Feed hopper capital cost = ($3,291 + $583 + $158) $4,022

Processing equipment—rougher jig (p. 37)

20 yd³/h

Base cost	$Y_C = 6{,}403.82(20)^{0.595}$	=	$38,067
Equipment price	$38,067 × 0.62 × 1.005	=	$23,720
Installation labor	$38,067 × 0.12 × 1.025	=	4,682
Construction materials	$38,067 × 0.26 × 1.015	=	10,046
Rougher factor	F_R	=	0.531

Rougher jig capital cost = [($23,720 + $4,682 + $10,046) × 0.531] $20,416

Processing equipment—cleaner jigs (p. 37)

2 at 5 yd³/h

Base cost	$Y_C = 6{,}403.82(5)^{0.595}$	=	$16,685
Equipment price	$16,685 × 0.62 × 1.005	=	$10,396
Installation labor	$16,685 × 0.12 × 1.025	=	2,052
Construction materials	$16,685 × 0.26 × 1.015	=	4,403

Cleaner jigs capital cost = [$10,396 + $2,052 + $4,403) × 2] $33,702

Processing equipment—final jig (p. 37)

0.2 yd³/h

Base cost	$Y_C = 6{,}403.82(0.2)^{0.595}$	=	$2,458
Equipment price	$2,458 × 0.62 × 1.005	=	$1,532
Installation labor	$2,458 × 0.12 × 1.025	=	302
Construction materials	$2,458 × 0.26 × 1.015	=	649

Final jig capital cost = ($1,532 + $302 + $649) $2,483

Processing equipment—sluice (p. 38)

50 yd³/h

Base cost	$Y_C = 113.57(50)^{0.567}$	=	$1,044
Construction labor	$1,044 × 0.61 × 1.025	=	$653
Construction materials	$1,044 × 0.39 × 1.015	=	413

Sluice capital cost = ($653 + $413) $1,066

Processing equipment—sluice (p. 38)

20 yd³/h

Base cost	$Y_C = 113.57(20)^{0.567}$	=	$621
Construction labor	$621 × 0.61 × 1.025	=	$388
Construction materials	$621 × 0.39 × 1.015	=	246

Sluice capital cost = ($388 + $246) $634

Processing equipmment—trommel (p. 41)

100 LCY/h

Base cost . . . . $Y_C = 7{,}176.21(100)^{0.559}$ = $94,166

- Equipment price . . . . $94,166 × 0.64 × 1.005 = $60,568
- Installation labor . . . . $94,166 × 0.26 × 1.025 = 25,095
- Construction materials . . . . $94,166 × 0.10 × 1.015 = 9,558

Trommel capital cost = ($60,568 + $25,095 + $9,558) $95,221

Supplemental—main building (p. 43)

1,680 ft²
Cement floor
Plumbing added

Base cost . . . . $Y_C = 34.09(1{,}680)^{0.907}$ = $28,707

- Equipment . . . . $28,707 × 0.25 × 1.005 = $7,213
- Construction labor . . . . $28,707 × 0.34 × 1.025 = 10,004
- Construction materials . . . . $28,707 × 0.41 × 1.015 = 11,946

Cement floor factor . . . . $F_C = 1.035(1{,}680)^{0.008}$ = 1.098
Plumbing factor . . . . $F_P = 1.013(1{,}680)^{0.002}$ = 1.028

Main building capital cost = [($7,213 + $10,004 + $11,946) × 1.098 × 1.028] $32,918

Supplemental—sheds (p. 43)

2 at 216 ft² each

Base cost . . . . $Y_C = 34.09(216)^{0.907}$ = $4,467

- Equipment . . . . $4,467 × 0.25 × 1.005 = $1,122
- Construction labor . . . . $4,467 × 0.34 × 1.025 = 1,557
- Construction materials . . . . $4,467 × 0.41 × 1.015 = 1,859

Shed capital costs = [($1,122 + $1,557 + $1,859) × 2] $9,076

Supplemental—employee housing (p. 44)

100 LCY/h pay gravel
250 LCY/h overburden
350 LCY/h total
Used trailers

Base cost . . . . $Y_C = 7{,}002.51(350)^{0.418}$ = $81,035

- Equipment . . . . $81,035 × 0.90 × 1.005 = $73,296
- Construction labor . . . . $81,035 × 0.07 × 1.025 = 5,814
- Construction materials . . . . $81,035 × 0.03 × 1.015 = 2,468

Used trailer factor . . . . F_U = 0.631

Employee housing capital cost = [($73,296 + $5,814 + $2,468) × 0.631] $51,476

Supplemental—generators (p. 45)

100-LCY/h mill feed

Base cost	$Y_C = 1{,}382.65(100)^{0.604}$	=	$22,321
Equipment	$22,321 × 0.75 × 1.005	=	$16,824
Construction labor	$22,321 × 0.19 × 1.025	=	4,347
Construction materials	$22,321 × 0.06 × 1.015	=	1,359

Generator capital cost = ($16,824 + $4,347 + $1,359) **$22,530**

Supplemental—pumps (p. 46)

100-LCY/h mill feed
80-ft head

Water consumption (p. 47) = $94.089(100)^{0.546}$ = 1,163 gpm

Base cost	$Y_C = 63.909(1{,}163)^{0.618}$	=	$5,013
Equipment	$5,013 × 0.70 × 1.005	=	$3,527
Installation labor	$5,013 × 0.08 × 1.025	=	411
Construction materials	$5,013 × 0.22 × 1.015	=	1,120
Head factor	$F_H = 0.125(80)^{0.637}$	=	2.038

Pump capital cost = [($3,527 + $411 + $1,120) × 2.038] **$10,308**

Supplemental—settling ponds (p. 47)

1,163 gpm

Base cost	$Y_C = 3.982(1{,}163)^{0.952}$	=	$3,300
Construction labor	$3,300 × 0.75 × 1.025	=	$2,537
Fuel and lubrication	$3,300 × 0.13 × 0.991	=	425
Equipment parts	$3,300 × 0.12 × 1.005	=	397

Settling pond capital cost = ($2,537 + $425 + $397) **$3,360**

CAPITAL COST SUMMARY FORM

Item	Cost
Exploration:	
Method 1 cost	$
Method 2 cost	94,485
Development:	
Access roads	88,180
Clearing	9,725
Preproduction overburden removal:	
Bulldozers	
Draglines	
Front-end loaders	
Rear-dump trucks	
Scrapers	12,077
Mine equipment:	
Backhoes	266,792
Bulldozers	69,022
Draglines	
Front-end loaders	272,207
Rear-dump trucks	
Scrapers	200,817
Processing equipment:	
Conveyors	16,115
Feed hoppers	4,022
Jig concentrators	56,601
Sluices	1,700
Spiral concentrators	
Table concentrators	
Trommels	95,221
Vibrating screens	
Supplemental:	
Buildings	41,994
Camp	51,476
Generators	22,530
Pumps	10,308
Settling ponds	3,360
Subtotal	1,316,632
Contingency (10%)	131,663
Total	1,448,295

Figure A-2.—Capital cost summary form completed for example estimation.

OPERATING COSTS

Overburden removal—scrapers (p. 52)

250 LCY/h
3,000-ft average haul distance
+8% average haul gradient plus rolling resistance
Used equipment

Equipment	$Y_E = 0.325(250)^{-0.210}$	=	$0.102/LCY
Parts	$0.102 × 0.18 × 1.005	=	$0.018
Fuel and lubrication	$0.102 × 0.48 × 0.991	=	0.049
Tires	$0.102 × 0.34 × 0.939	=	0.033
			$0.100
Labor	$Y_L = 12.01(250)^{-0.930}$	=	$0.071/LCY
Labor	$0.071 × 1.00 × 1.025	=	$0.073
Distance factor	$F_D = 0.01947(3{,}000)^{0.577}$	=	1.975
Gradient factor	$F_G = 0.776e^{[0.047(8)]}$	=	1.130
Used equipment factor	$U_e = 1.096(250)^{-0.006}$	=	1.060
	$U_l = 0.845(250)^{0.034}$	=	1.019

Overburden removal cost ....[(0.100 × 1.060) + (0.073 × 1.019)] × 1.975 × 1.130 = $0.403/LCY

Annual scraper operating cost = $0.403/LCY × 375,000 LCY/a $151,125

Mining—backhoes (p. 53)

Pay gravel excavation
100 LCY/h
80% maximum digging depth
Medium-hard digging difficulty

Equipment	$Y_E = 8.360(100)^{-1.019}$	=	$0.077/LCY
Parts	$0.077 × 0.38 × 1.005	=	$0.029
Fuel and lubrication	$0.077 × 0.62 × 0.991	=	0.047
			$0.076
Labor	$Y_L = 17.53(100)^{-1.009}$	=	$0.168/LCY
Labor	$0.168 × 1.00 × 1.025	=	$0.172
Digging depth factor	$F_D = 0.09194(80)^{0.608}$	=	1.320
Digging difficulty factor	F_H	=	1.500

Backhoe mining cost = [(0.076 + 0.172)] × 1.320 × 1.500 = $0.491/LCY

Annual backhoe operating cost = $0.491/LCY × 150,000 LCY/a $73,650

Mining—front-end loaders (p. 56)

Pay gravel haulage
100 LCY/h total
Two 50-LCY/h loaders
800-ft average haul distance
+6% average haul gradient plus rolling resistance

Equipment .... $Y_E = 0.407(50)^{-0.225}$ = $0.169/LCY

Parts .... $0.169 × 0.22 × 1.005 = $0.037
Fuel and lubrication .... $0.169 × 0.46 × 0.991 = 0.077
Tires .... $0.102 × 0.32 × 0.939 = 0.051
$0.165

Labor .... $Y_L = 13.07(50)^{-0.936}$ = $0.336/LCY

Labor .... $0.336 × 1.00 × 1.025 = $0.344

Distance factor .... $F_D = 0.023(800)^{0.616}$ = 1.413
Gradient factor .... $F_G = 0.877e^{[0.046(6)]}$ = 1.156

Pay gravel transportation cost = (0.165 + 0.344) × 1.413 × 1.156 = $0.831/LCY

Annual front-end loader operating cost = $0.831/LCY × 150,000 LCY/a $124,650

Processing—conveyors (p. 59)

70 yd³/h

Equipment .... $Y_E = 0.218(70)^{-0.561}$ = $0.020/yd³

Parts .... $0.020 × 0.72 × 1.005 = $0.014
Electricity .... $0.020 × 0.24 × 1.029 = 0.005
Lubrication .... $0.020 × 0.04 × 0.991 = 0.001
$0.020

Labor .... $Y_L = 0.250(70)^{-0.702}$ = $0.013/yd³

Labor .... $0.013 × 1.00 × 1.025 = $0.013

Conveyor operating cost = (0.020 + 0.013) = $0.033/yd³

Annual conveyor operating cost = $0.033/yd³ × 105,000 yd³/a $3,465

Processing—feed hoppers (p. 60)

100 LCY/h total

Equipment .... $Y_E = 0.033(100)^{-0.344}$ = $0.007/LCY

Parts .... $0.007 × 0.88 × 1.005 = $0.006
Electricity .... $0.007 × 0.06 × 1.029 = 0.0004
Lubrication .... $0.007 × 0.06 × 0.991 = 0.0004
$0.007

Labor .... $Y_L = 0.017(100)^{-0.295}$ = $0.004/LCY

Labor .... $0.004 × 1.00 × 1.025 = $0.004

Feed hopper operating cost = (0.007 + 0.004) = $0.011/LCY

Annual feed hopper operating cost = $0.011/LCY × 150,000 LCY/a $1,650

Processing—rougher jig (p. 61)

20 yd³/h

Equipment	$Y_E = 0.113(20)^{-0.328}$	=	$0.042/yd³
Parts	$0.042 × 0.40 × 1.005	=	$0.017
Electricity	$0.042 × 0.34 × 1.029	=	0.015
Lubrication	$0.042 × 0.26 × 0.991	=	0.011
			$0.043
Supplies	$Y_S = 0.002(20)^{-0.184}$	=	$0.001/yd³
Industrial materials	$0.001 × 1.00 × 1.003	=	$0.001
Labor	$Y_L = 3.508(20)^{-1.268}$	=	$0.079/yd³
Labor	$0.079 × 1.00 × 1.025	=	$0.081
Rougher service factor	F_R	=	0.344

Rougher jig operating cost = (0.043 + 0.001 + 0.081) × 0.344 = $0.043/yd³

Annual rougher jig operating cost = $0.043/yd³ × 30,000 yd³/a $1,290

Processing—cleaner jigs (p. 61)

2 at 5 yd³/h

Equipment	$Y_E = 0.113(5)^{-0.328}$	=	$0.067/yd³
Parts	$0.067 × 0.40 × 1.005	=	$0.027
Electricity	$0.067 × 0.34 × 1.029	=	0.023
Lubrication	$0.067 × 0.26 × 0.991	=	0.017
			$0.067
Supplies	$Y_S = 0.002(5)^{-0.184}$	=	$0.001/yd³
Industrial materials	$0.001 × 1.00 × 1.003	=	$0.001
Labor	$Y_L = 3.508(5)^{-1.268}$	=	$0.456/yd³
Labor	$0.456 × 1.00 × 1.025	=	$0.467

Cleaner jig operating cost = (0.067 + 0.001 + 0.467) = $0.535/yd³

Annual cleaner jig operating cost = $0.535/yd³ × 15,000 yd³/a $8,025

Processing—final jig (p. 61)

0.2 yd^3/h

Equipment	$Y_E = 0.113(0.2)^{-0.328}$	=	$0.192/$yd^3$
Parts	$0.192 × 0.40 × 1.005	=	$0.077
Electricity	$0.192 × 0.34 × 1.029	=	0.067
Lubrication	$0.192 × 0.26 × 0.991	=	0.049
			$0.193
Supplies	$Y_S = 0.002(0.2)^{-0.184}$	=	$0.003/$yd^3$
Industrial materials	$0.003 × 1.00 × 1.003	=	$0.003
Labor	$Y_L = 3.508(0.2)^{-1.268}$	=	$26.999/$yd^3$
Labor	$26.999 × 1.00 × 1.025	=	$27.674

Final jig operating cost = (0.193 + 0.003 + 27.674) = $27.870/$yd^3$

Annual final jig operating cost = $27.870/$yd^3$ × 300 yd^3/a $8,361

Processing—sluices (p. 62)

50 yd^3/h

Labor	$Y_L = 0.377(50)^{-0.636}$	=	$0.031/$yd^3$
Labor	$0.031 × 1.00 × 1.025	=	$0.032

Sluice operating cost = $0.032/$yd^3$

Annual sluice operating cost = $0.032/$yd^3$ × 75,000 yd^3/a $2,400

Processing—Sluices (p. 62)

20 yd^3/h

Labor	$Y_L = 0.377(20)^{-0.636}$	=	\$0.056/yd^3
Labor	\$0.056 × 1.00 × 1.025	=	\$0.057

Sluice operating cost = \$0.057/yd^3

Annual sluice operating cost = \$0.057/yd^3 × 30,000 yd^3/a \$1,710

Processing—Tailings removal—bulldozers (p. 65)

100 LCY/h
400-ft average haul distance
−8% average haul gradient

Equipment	$Y_E = 0.993(100)^{-0.430}$	=	\$0.137/LCY
Parts	\$0.137 × 0.47 × 1.005	=	\$0.065
Fuel and lubrication	\$0.137 × 0.53 × 0.991	=	0.072
			\$0.137
Labor	$Y_L = 14.01(100)^{-0.945}$	=	\$0.180/LCY
Labor	\$0.180 × 1.00 × 1.025	=	\$0.185
Distance factor	$F_D = 0.00581(400)^{0.904}$	=	1.307
Gradient factor	$F_G = 1.041e^{[0.015(-8)]}$	=	0.923
Used equipment factor	$U_e = 1.206(100)^{-0.013}$	=	1.136
	$U_l = 0.967(100)^{0.015}$	=	1.036

Tailings removal cost = [(0.137 × 1.136) + (0.185 × 1.036)] × 1.307 × 0.923 = \$0.419/LCY

Annual bulldozer operating cost = \$0.419/LCY × 150,000 LCY/a \$62,850

Processing—trommels (p. 70)

100 yd^3/h

Equipment	$Y_E = 0.217(100)^{-0.403}$	=	\$0.034/yd^3
Parts	\$0.034 × 0.63 × 1.005	=	\$0.022
Electricity	\$0.034 × 0.26 × 1.029	=	0.009
Lubrication	\$0.034 × 0.11 × 0.991	=	0.004
			\$0.035
Labor	$Y_L = 0.129(100)^{-0.429}$	=	\$0.018/yd^3
Labor	\$0.018 × 1.00 × 1.025	=	\$0.018

Trommel operating cost = (\$0.035 + \$0.018) = \$0.053/yd^3

Annual trommel operating cost = \$0.053/yd^3 × 150,000 yd^3/a \$7,950

Supplemental—housing (p. 72)

100 LCY/h pay gravel
250 LCY/h overburden
350 LCY/h total

Supplies	$Y_S = 1.445(350)^{-0.583}$	=	\$0.047/LCY
Fuel	\$0.047 × 0.05 × 0.991	=	\$0.002
Industrial materials	\$0.047 × 0.95 × 1.003	=	0.045
			\$0.047

Housing operating cost = \$0.047/LCY

Annual housing operating cost = \$0.047/LCY × 525,000 LCY/a \$24,675

Supplemental—lost time and general services (p. 74)

100-yd³/h mill feed

Equipment	$Y_E = 0.142(100)^{0.004}$	=	\$0.145/LCY
Fuel	\$0.145 × 0.53 × 0.991	=	\$0.076
Parts	\$0.145 × 0.47 × 1.005	=	0.068
			\$0.144
Labor	$Y_L = 2.673(100)^{-0.524}$	=	\$0.239/LCY
Labor	\$0.239 × 1.00 × 1.025	=	\$0.245

Lost time and general service cost = (\$0.144 + \$0.245) = \$0.389/LCY

Annual lost time and general service cost = \$0.389/LCY × 675,000 LCY/a \$262,575

Supplemental—pumps (p. 75)

100 yd³/h mill feed
1,163 gpm
80-ft head

Equipment	$Y_E = 0.007(1{,}163)^{0.713}$	=	\$1.074/h
Fuel and lubrication	\$1.074 × 0.59 × 0.991	=	\$0.628
Parts	\$1.074 × 0.41 × 1.005	=	0.443
			\$1.071
Labor	$Y_L = 0.004(1163)^{0.867}$	=	\$1.819/h
Labor	\$1.819 × 1.00 × 1.025	=	\$1.864
Head factor	$H_e = 0.091(80)^{0.735}$	=	2.279
	$H_l = 0.054(80)^{0.893}$	=	2.739

Pump operating cost = [(\$1.071 × 2.279) + (\$1.864 × 2.739)] = \$7.546/h

Annual pump operating cost = \$7.546/h × 1,500 h/a \$11,319

OPERATING COST SUMMARY FORM

Item	Annual cost
Overburden removal:	
Bulldozers	$
Draglines	
Front-end loaders	
Rear-dump trucks	151,125
Scrapers	
Mining:	
Backhoes	73,650
Bulldozers	
Draglines	
Front-end loaders	124,650
Rear-dump trucks	
Scrapers	
Processing:	
Conveyors	3,465
Feed hoppers	1,650
Jig concentrators	17,676
Sluices	4,110
Spiral concentrators	
Table concentrators	
Tailings removal:	
Bulldozers	62,850
Draglines	
Front-end loaders	
Rear-dump trucks	
Scrapers	
Trommels	7,950
Vibrating screens	
Supplemental:	
Employee housing	24,675
Lost time and general services	262,575
Pumps	11,319
Subtotal	745,695
Contingency (10%)	74,570
Total	820,265

Cost per cubic yard pay gravel = total annual cost divided by pay gravel mined per year.

$820,265/150,000 LCY/a = $5.47/LCY

Final cost per cubic yard pay gravel $5.47

Figure A-3.—Operating cost summary form completed for example estimation.

☆U.S. GOVERNMENT PRINTING OFFICE: 1987—196-191/70448

Placer Sample Forms

Placer Sampling and Processing Forms

Available from Denver

Form No.	Release Date	Title & Usage
3842-1	May, 1986	**Placer Sample Field Record.** Use at each sample site while collecting your placer sample. Two sided. Space to draw a diagram of the outcrop on the back.
3842-2	December, 1977	**Placer Sample Processing Record.** Use as you process each sample from concentration to amalgamation. Single sided.
3842-3	December, 1977	**Placer Sample Summary Sheet.** Use to keep track of each sample in a project. Single sided.

Provided in this notebook

Title & Usage

Placer Sample Bucket Tally. Use while collecting sample to determine volume in loose cubic yards (LCY), total weight, and volume of oversize (larger than 1/4 inch). Provides essential information to your calculations. Two sided. Displays volumes of partially filled buckets on the back. BLM-NTC-300-3K9-1

Sample Site Photo Placard. Use to establish before and after photographs of each sample site. Use dark, wide felt marker. Enter sample number and date, pose in photo before you collect the sample. After collecting the sample, print "AFTER" below the date, and photograph next to sample channel. Duplicate on green or blue paper to reduce sun glare. White paper will produce a washed out, illegible photograph. BLM-NTC-300-3K9-2

Bucket Volume & Pan Factors. Can be cut out and punched for field notebook, or glued directly to notebook pages.

Description and Character of Placer Gold Recovered. Useful when examining placer concentrates under a binocular microscope.

Precious Metal Placer Sample Report. A reduction of an oversize form developed by J.R. Evans, CA SO. Similar to 3842-3. Two sided, explanations on rear.

Example Assay Instructions for Placer Concentrates. Standard assay instructions for requesting mercury amalgamations. Will help you prevent the receipt of bad data or erroneously formatted data. (You DO NOT want results in ounces per ton!)

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Placer Sample Bucket Tally

Sample Number:______________________________ Date:______________________

Project:______________________________ Sampled by:______________________

Bucket tare weights: 5 gal.________ $3^1/_2$ gal________ Processed by:______________________

#	Weight, Lb.	Fill from top	Est. % H_2O	Vol LCY	Size: 5	$3^1/_2$	#	Weight, Lb.	Filll from top	Est. % H_2O	Vol LCY	Size: 5	$3^1/_2$
1							26						
2							27						
3							28						
4							29						
5							30						
6							31						
7							32						
8							33						
9							34						
10							35						
11							36						
12							37						
13							38						
14							39						
15							40						
16							41						
17							42						
18							43						
19							44						
20							45						
21							46						
22							47						
23							48						
24							49						
25							50						
		Subtotals							Subtotals				

Total Weight:______________ Total Volume LCY:____________________

Buckets of oversize:______________ (5 gallon)

BUCKET VOLUME: 5 Gallon Plastic Ropak Brand
Calculated volumes of partially filled buckets.
41 struck buckets equal 1.02 cubic yards.
Calculations +/- 0.0005 and assume cylindrical shape.

Inches From Bottom	Inches From Top	Volume in Loose Cubic Feet	Volume in Loose Cubic Yards
14	0	0.6685	0.0248
13	1	0.6208	0.0230
12	2	0.5730	0.0212
11	3	0.5252	0.0195
10	4	0.4775	0.0177
9	5	0.4298	0.0159
8	6	0.3820	0.0141
7	7	0.3343	0.0124
6	8	0.2865	0.0106
5	9	0.2387	0.0088
4	10	0.1910	0.0071
3	11	0.1433	0.0053
2	12	0.0955	0.0035
1	13	0.0477	0.0018

(MWS, 4/93)

BUCKET VOLUME: $3^1/_2$ Gallon Plastic Letica brand
Calculated volumes of partially filled buckets.
58 struck buckets equal 1.003 cubic yards.
Calculations +/- 0.0005 and assume cylindrical shape

Inches From Bottom	Inches From Top	Volume in Loose Cubic Feet	Volume in Loose Cubic Yards
10.5	0	0.4679	0.0173
9.5	1	0.4234	0.0157
8.5	2	0.3788	0.0140
7.5	3	0.3342	0.0124
6.5	4	0.2896	0.0107
5.5	5	0.2451	0.0091
4.5	6	0.2005	0.0074
3.5	7	0.1560	0.0058
2.5	8	0.1114	0.0041
1.5	9	0.0669	0.0025
0.5	10	0.0223	0.0008

(MWS, 4/93)

Sample No. ______________________

Date: ______________________

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BUCKET VOLUME: 5 Gallon Plastic Ropak Brand
Calculated volumes of partially filled buckets.
41 struck buckets equal 1.02 cubic yards.
Calculations +/- 0.0005 and assume cylindrical shape.

Inches From Bottom	Inches From Top	Volume in Loose Cubic Feet	Volume in Loose Cubic Yards
14	0	0.6685	0.0248
13	1	0.6208	0.0230
12	2	0.5730	0.0212
11	3	0.5252	0.0195
10	4	0.4775	0.0177
9	5	0.4298	0.0159
8	6	0.3820	0.0141
7	7	0.3343	0.0124
6	8	0.2865	0.0106
5	9	0.2387	0.0088
4	10	0.1910	0.0071
3	11	0.1433	0.0053
2	12	0.0955	0.0035
1	13	0.0477	0.0018

(MWS, 4/93)

BUCKET VOLUME: 3$^1/_2$ Gallon Plastic Letica brand. Calculated volumes of partially filled buckets. 58 struck buckets equal 1.003 cubic yards. Calculations +/- 0.0005 and assume cylindrical shape

Inches From Bottom	Inches From Top	Volume in Loose Cubic Feet	Volume in Loose Cubic Yards
10.5	0	0.4679	0.0173
9.5	1	0.4234	0.0157
8.5	2	0.3788	0.0140
7.5	3	0.3342	0.0124
6.5	4	0.2896	0.0107
5.5	5	0.2451	0.0091
4.5	6	0.2005	0.0074
3.5	7	0.1560	0.0058
2.5	8	0.1114	0.0041
1.5	9	0.0669	0.0025
0.5	10	0.0223	0.0008

(MWS, 4/93)

PAN FACTORS (After Wells, 1989)

Pan Diameter, Inches	Volume in Loose Cubic Yards	Volume in Bank Cubic Yards	Pans Per Bank Cubic Yard
12	0.0032 (est.)	0.00250	400
14	0.0051 (est.)	0.00400	250
16	0.0072 (Wells)	0.00556	180

Note: Pan factors cannot be compared directly to bucket volume. They are rule-of-thumb conversions to bank cubic yards based on examiners' experience and are affected by variations from the "average" swell rate, heaping of materials in the pan, and actual pan volume. (Not all equal diameter pans have equal volumes.) Pan factors should be used with great care, as major volumetric errors may occur. These errors may be hard to spot and may cause skewed values. The best approach is to measure your own pan volume and estimate swell factors for each sample.
Better yet: collect big samples.

(MWS, 4/93)

Cut out these tables, glue on field note pages, and put in your Handbook for Mineral Examiners, H-3890-1 They should just fit (barely) on Rite-in-the-Rain[R] field notebook paper.

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Title of Mineral Study________________________ Case Number__________ Page ___ of ___
Organization and Office________________________ Analysis Performed at:____________
Examiner's Name(s) and Signatures:________________________________
Analysis Method: Binocular Microscopic Examination [] Other [] (specify)________________

DESCRIPTION AND CHARACTER OF PLACER GOLD RECOVERED							
Sample Number	Color and Staining	Surface Texture	Associated Minerals	Shape	Size	Adhered Rock or Mineral Fragments	Remarks, Including Evidence of Salting

Coarse gold = 10 mesh (2.00mm or 0.079 in.) Medium gold = -10 to +20 mesh (.84mm or .033 in.)
Fine gold = -20 to +40 mesh (0.420mm or 0.017 in.) Very fine gold = -40 mesh.

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PRECIOUS METAL PLACER SAMPLE REPORT FORM

Study Title ______________________ **Field Exam Date(s)** __________ **Case File No.** ______ **Page** ___ **of** ___

Organization and Office ______________________ **Location Where Analyses Performed** __________ **Date** ______

Analyst's Name(s) and Signature(s) ______________________ **Date** ______

GOLD DATA 4/				
Price ($/T. oz.)	Fineness (Assume 900)	Cents/Mg. 31.104 mg per Troy oz. 13/	Mg. / yd.³ 14/	Cents /yd.³ 15/

GENERAL DATA					SAMPLE AND SAMPLE SITES																CONCENTRATES (lbs.) AND SAMPLE GOLD RECOVERY (Mg)							REMARKS
Smpl. No.	Smpl. Type 1/	Mining Claim Name and Location	Date Smpl. Taken	Rock Unit and Age	Dimensions (ft.) 2/	Volume (ft.³) Bank 3/	Volume (ft.³) Loose 3/	Volume (ft.³) Boulders Deposit Grizzly 4/	Volume Reduction for ___ % Swell 5/ 6/	Volume Total 6/ ft.³	Volume Total yd.³	Volume Adjusted (A) 7/ ft.³	Volume Adjusted (A) yd.³	Weight Factor 8/ ft.³	Weight Factor yd.³	Weight of Sample (lbs.) 9/ - Bould.	Weight of Sample (lbs.) Bould.	Weight of Sample (lbs.) Total	Volume Adjusted (B) 10/ ft.³	Volume Adjusted (B) yd.³	Weight of Concentrates (lbs.) + 8 mesh	Weight of Concentrates (lbs.) - 8 mesh	Weight of Concentrates (lbs.) Total	Gravity Primary Equipment 11/ Mg. Primary Recovery	Gravity Secondary Equipment 12/ Mg. Secondary Recovery	Amalgamation Mg. Prim. Recov.	Amalgamation Mg. Secon. Recov.	

General notes and remarks ....

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(See following pages for explanation of footnotes.)

JHE 8/7/89

EXPLANATION OF FOOTNOTES FOR PRECIOUS METAL PLACER SAMPLE REPORT FORM

1/ Backhoe, H; Pick and shovel channel, C; Bedrock, B; Pan, P; Drill, D; Loader, L.

2/ If dimensions of cut can be taken, volume can be computed and the weight of the material will then allow computation of a weight per unit volume directly.

3/ Plastic 5 gallon bucket contains 0.67 ft.^{3}, and 27 ft.^{3}/yd.^{3} ÷ 0.67 ft.^{3} /bucket = 40.3 buckets / yd.^{3} (w/o swell factor).

4/ Estimated by eye and an average taken. Deposit refers to deposit as a whole, grizzly refers to those boulders which will not be moved and/or pass grizzly and not go through processing circuit.

5/ 10% ± is common for many sand and gravel deposits.

6/ Will equal bank volume where dimensions are available. IF volume is loose with a 10% swell factor and a loose volume of say 23.2 ft.^{3} then 23.2 x 0.90 = 20.9 volume reduction. Add on say 8.9 ft.^{3} of boulder volume to make 29.8 ft.^{3} or 1.10 yd.^{3} for total volume.

7/ Use this column where you wish to adjust for material not mined and/or passing over grizzly. Figure this material in mining and/or operating cost section of report (see 4/).

8/ Figure as in 2/ or use cubic foot box.

9/ Use where only weight of sample is available and a unit weight has been determined by cubic foot box (see 2/). Say there are 582 lbs. of sample without boulders and 119 lbs. of boulders (estimated 0.70 ft.^{3} x 170 lbs./ft.^{3} = 119 lbs.) and unit weight is 111 lbs./ft.^{3} (3,004 lbs./yd.^{3}): 582 lbs. + 119 lbs. ÷ 111 lbs./ft.^{3} = 6.3 ft.^{3} or 0.233 yd.^{3} of sample.

10/ If you want to use weight only for boulder factor (BF) adjustment use this column for reporting results by following formula:

$$\text{Smpl. vol. (yd.}^3) = \frac{\text{sample wt.}}{1 - \text{BF as decimal}} \times \frac{1}{\text{wt./yd.}^3}$$

$$V\ \text{yd.}^3 = \frac{1000\ \text{lbs.}}{(1-0.20)} \times \frac{1}{3000\ \text{lbs./yd.}^3} = 0.413\ \text{yd}^3.$$

11/ List equipment used.

12/ List equipment used.

13/ With gold at \$400/oz.: $\frac{\$400 \times 900 \times 0.1}{31{,}104\ \text{mg.}} = 1.157$ cts./mg.

14/ With say 629 mg. in 1.10 yd.^{3} ∴ 629 mg./1.10 yd.^{3} = x mg./one yd.^{3} or 1.10 x = 629 or 572 mg./yd.^{3}.

15/ See 1/ & 14/ 572 mg./yd.^{3} x 1.157 cts./mg. = 662 cts./yd.^{3} or \$6.62/yd.^{3}.

JRE 8-7-89

Example Assay Instructions for Placer Concentrates

Amalgamation only

Assay Instructions

Enclosed are six black sand concentrates for mercury amalgamation. You should shortly receive a purchase order to cover the cost. Please do not begin work until it arrives.

The following needs to be done with each sample.

1. Mercury amalgamation:

   A. Report sample dry weight.
   B. Report gold and silver concentration in total milligrams.
   C: Please do not report concentrations in ounces per ton of concentrates.

2. Fineness determination.

3. Return pulps and results to:

   Matt Shumaker
   Bureau of Land Management
   National Training Center
   9828 North 31 Avenue
   Phoenix, AZ 85051.

Please give me a call at (602) 906-5526 if you have any questions.

Amalgamation Followed by Bottle Roll Cyanide

Enclosed are 5 black sand concentrates for mercury amalgamation. You should shortly receive a purchase order from BLM Safford District to cover the cost. Please do not begin work until it arrives.

The following needs to be done with each sample.

1. Mercury amalgamation. Report sample dry weight, and gold and silver from amalgamation in total milligrams. Please do not report results in ounces per ton of concentrates, or as a "head assay."

2. Fineness determination.

3. After amalgamation, Cyanide bottle roll. Report gold and silver from cyanide bottle roll in total milligrams. Please do not report results in ounces per ton of concentrates, or as a "head assay."

4. Return pulps and results to:

   Larry Thrasher
   Bureau of Land Management
   Safford District.
   711 14th Avenue
   Safford, AZ 85546.

Please give Matt Shumaker a call at (602) 906-5526 if you have any questions about sample processing.

Please call Larry Thrasher at (520) 428-4040 for purchase order questions.

Metal Detectors

Treasure Hunter's GLOSSARY

(With special acknowledgment to *Western and Eastern Treasures' Buyers Guide*)

AIR TEST — An evaluation of a metal detector's performance based on response to various targets passed beneath the searchcoil in open air. Although demonstrating sensitivity, such a test is unreliable as an indicator of depth of detection of in-ground targets.

ALL-METAL MODE — A type of operation in which a metal detector is fully responsive to both ferrous and non-ferrous targets. Usually including elimination of ground effect problems, it is also referred to as normal mode.

AUDIO ID — A feature which produces differences in signal tone (i.e., pitch or frequency), enabling the operator to determine the probable identity or category of the target by its characteristic sound.

AUTOMATIC TUNING — A feature which constantly compensates for electronic or environmental changes which would otherwise cause a metal detector to drift from its optimum tuning level. In order to prevent a discriminate-mode problem known as "overshoot," a slow retuning speed is generally preferred.

BACKREADING — A false signal occurring in discriminate mode when a rejected target comes into close proximity (1" or so) of actual contact with the searchcoil of a metal detector. Such signals are harsh, abrupt, and readily distinguished from normal, good-target signals. Also, by raising the searchcoil slightly and re-checking, suspected backreading signals can be eliminated; good signals, although fainter, will still be received after the coil is raised.

BENCH TEST — A type of air test in which various metal detector modes, features, and functions are evaluated. This term is often used to refer to discriminate-mode testing, with assorted ferrous and non-ferrous samples being checked in a metal-free area.

BLACK SAND — An intensely negative form of iron oxide. Also called magnetite, it is often associated with gold-bearing mineral deposits. Metal-detector operation can be difficult in locations where there are heavy concentrations of black sand.

CACHE — An intentionally concealed hoard of valuables.

CHATTER — Undesirable signals produced by surface trash located by a motion discriminator. Such signals may be short, choppy, sputtery, or scratchy, and they are readily distinguished from the strong, smooth, sustained signals produced by good targets. Chatter may also result from mineralization if too much sensitivity is used.

COIN DEPTH METER — A feature in which target-signal intensity is translated into increments of depth displayed on a meter scale. Calibrated to coin-size targets, such a meter provides an approximate indication of depth; very large or small targets will be interpreted with less accuracy.

CONDUCTIVITY — The measure of a metal target's ability to allow eddy current generation on its surface. These currents then produce an inductive imbalance in the searchcoil, thus causing a signal.

DEPTH PENETRATION — The extent of a metal detector's ability to transmit an electromagnetic field into a ground mineral matrix and produce a target signal.

DETUNING — Adjusting the threshold into the null or less sensitive portion of the tuning. Also a method of narrowing a target signal's width by retuning to audio threshold over the target response area, thereby aiding pinpointing.

DISCRIMINATE MODE — A type of operation in which a metal detector is adjusted to reject unwanted metal targets such as nails, foil, bottle caps, and pulltabs. This is achieved by adjustable circuitry which ignores or nulls out response from targets within or below a specific conductivity range. Often simply called "discrimination," with metal detectors capable of such performance being referred to as "discriminators."

DRIFT — A loss of threshold tuning stability. Drift may be caused by changes in temperature, humidity, matrix mineralization, battery condition and/or metal detector design. High-quality instruments are frequency-stable and feature automatic or manual-drift correction circuitry.

ELLIPTICAL — Oval or elongated search coil in configuration, rather than uniformly circular.

FAINT SIGNAL — A weak, but audible, response characteristic of a very small, or deeply buried target.

FALSE SIGNAL — An audio indication created by backreading, overshoot, ground voids, hot rocks or other mineralization, wet vegetation, interference, or some other source of disruption. Although varying widely in quality and intensity, such signals are usually easy to distinguish from those of good targets.

FIELD TEST — An evaluation of a metal detector's performance based on random location of targets under varying, often adverse search conditions. Although lacking the control of air or bench-testing, such a test provides more useful data in terms of actual operation.

GRIDDING — A method of systematically searching with a metal detector. Following real or imaginary parallel lines approximately three feet apart, the operator scans back and forth across the area. When this search is completed, the procedure is repeated, this time at right angles to the original lines. In this way, complete coverage is assured.

GROUND BALANCE (PRESET) — A feature which eliminates the need to perform manual bround balance. Circuitry is internally adjusted during manufacture to provide optimum operation over average ground conditions.

GROUND EFFECT — Negative or positive mineralization which affects the tuning and performance of a metal detector. The greater the amount of mineralization, the greater the problem. In non-motion TR discriminate mode, especially, both sensitivity and discrimination compound ground effect as they are increased. In all-metal and motion-discriminate modes, ground effect can be largely or totally eliminated.

HOT ROCK — Any rock which is more highly mineralized than the matrix surrounding it. Because the metal detector is compensating for the level of mineralization of the matrix, the "hot rock" produces a positive response in motion-discriminate mode. In the all-metal mode, the "hot rock" will produce a negative response or null, however. By comparing signals received in the two modes, it is possible to identify these rocks as undesirable targets.

INTERFERENCE — Audio disruption or other operational difficulty resulting from the presence of power lines, radio transmitters, etc., in the area in which a metal detector is being used. Metal detectors having the same operating frequency can experience mutual interference or "cross-talk" in some instances. Interference can be reduced by lowering the sensitivity setting of the detector and maintaining a reasonable distance from the source of the disturbance.

MATRIX — The total volume of ground penetrated by the transmitted electromagnetic field of a metal detector's searchcoil. The matrix may contain varying amounts of metal, minerals, moisture, and other substances.

MINERALIZED GROUND — Any soil which contains conductive or non-conductive components which affect the tuning and performance of a metal detector.

MOTION DISCRIMINATE MODE — A type of operation in which both automatic or manual ground rejection and discrimination occur simultaneously while the searchcoil is kept in motion. A metal detector having this capability is referred to as a "motion discriminator."

NOTCH ACCEPT — A feature whereby all target responses are eliminated, except those of a chosen notch level (conductivity) and notch width.

NOTCH DISCRIMINATION — A type of discrimination mode which allows a desirable lower conductivity range of targets to be accepted within the rejection range of a standard discriminate control setting. For example, notch discrimination permits detection of nickels and small gold rings while maintaining rejection of aluminum pulltabs — something impossible to achieve with conventional discriminate mode operation. Notch discrimination also permits rejection of all metal targets except those within a specific conductivity range; thus, a metal detector with this capability can be adjusted to respond only to gold rings. For additional information, see the article "Notch Filter Discrimination — Knack of the Notch."

NOTCH LEVEL — A control which selects the target conductivity range to which the notch filter circuitry will respond. Sometimes called the "notch window."

NOTCH REJECT — A feature whereby targets of a specific notch level (conductivity) and notch width are eliminated, producing no audio response.

NUGGETSHOOTING — Using a metal detector to search for gold nuggets. The all-metal mode is employed in this type of detecting. Often a small diameter searchcoil is preferred because of its greater responsiveness to tiny targets.

NULL — The zone below audible threshold in metal detector t[illegible]ing. Also, the momentary drop or quiet response of threshold audio as the searchcoil passes over a rejected target in discriminate mode. Note: These concepts do not apply to "silent search" operation.

ORE SAMPLING — A type of air test in which ore samples are passed beneath a metal detector's searchcoil. Responses are checked against those of other samples of known content or conductivity, obtaining a comparative indication of the metal or mineral present in the ore. Such tests can be conducted in either non-motion discriminate mode (at the lowest possible setting) or all-metal mode (with the detector adjusted to null slightly on a known iron mineral sample).

OVERLAPPING — The practice of advancing a metal detector's searchcoil by a distance which is less than the physical diameter of the coil, so that a portion of the ground covered by the coil's first pass will be included in its second pass as well. By overlapping, 25% or so, an operator is far less likely to miss targets.

OVERSHOOT — A false signal occurring in non-motion discriminate mode when automatic tuning is used. A rejected target creates a null which is immediately compensated for by the tuning circuitry; as it does so, the searchcoil passes bey[illegible] the rejected target's influence, and excessive retuning occu[illegible] This causes a momentary signal increase prior to return to normal threshold. Overshoot is particularly troublesome when retune speed is too fast and/or searchcoil movement is too slow.

PINPOINTING — The practice of precisely locating a target prior to attempting to recover it. By approaching the target from several different directions and noting the point at which the signal is strongest, it is possible to determine its exact position. Detuning and probing are additional techniques in pinpointing.

REJECTION — Non-acceptance of a target in the discriminate mode, indicated either by a null in the audible threshold or no change in sound.

RETUNING — Manual or automatic correction of drift or any other undesirable change in tuning.

RF TWO-BOX — A radio frequency detector having its transmit and receive windings in separate housings positioned several feet apart, at opposite ends of a connecting shaft. Two-box detectors are designed for locating large, deeply buried objects; they are not suitable for locating targets such as coi[illegible] Effectively used by cache hunters, they are also employed[illegible] prospecting when seeking or mapping out ore bodies, and they have industrial applications as well.

SCRUBBING — A technique used in non-motion discriminate mode to reduce ground effect problems due to searchcoil elevation, and to increase depth of detection. The searchcoil is pressed against the surface while searching.

SCUFF PLATE — A protective plastic cover which is friction-fitted to the bottom of a metal detector's searchcoil, preventing abrasion damage without affecting performance in any way. A scuff plate is particularly recommended when scrubbing the searchcoil during searching.

SEARCHCOIL — A circular plastic housing containing single or multiple transmit and receive windings in a specific orientation or configuration. (See the Question/ Answer section of this booklet for an extensive discussion of searchcoil designs and operational characteristics.)

SEARCHCOIL SHIELDING — A metallic or conductive barrier between the searchcoil windings and plastic housing, designed to eliminate electrostatic interference from wet vegetation. Elimination of this disruptive "grass effect" can also be achieved by means of special circuitry.

SENSITIVITY — The measure or capacity of a metal detector to perceive changes in conductivity within the searchcoil detection pattern. Adjustment of a metal detector's sensitivity affects both its depth of detection and its reaction to ground mineralization. In discriminate mode, reducing sensitivity may actually improve performance by decreasing ground effect and interference.

SIGNAL — An audio response and/or meter indication alerting the operator that a target has been located by the metal detector.

SIGNAL WIDTH — The total distance of ground over which a signal is sustained during searchcoil movement across the target area.

SILENT SEARCH — A type of metal detector operation in which optimum tuning is maintained without audible threshold, and without loss of sensitivity. No sound is heard until a target is detected.

STABILITY — The ability or tendency of a metal detector to maintain optimum tuning despite changes in temperature, humidity, matrix mineralization, or other environmental factors contributing to drift.

SURFACE BLANKING — A feature which permits the operator of a metal detector to eliminate all target signals from objects on or near the surface. A blank depth control regulates the distance of target elimination, usually from 1" to 4." Surface blanking circuitry is based on interpretation of signal intensity and calibrated to coin-size targets; very large or small targets will be eliminated at correspondingly greater or lesser distances than the blank depth setting. The Tesoro Royal Sabre features surface blanking and depth control.

TARGET — Any object causing a signal indication from a metal detector.

TARGET MASKING — An undesirable effect in discriminate mode whereby the negative response of a rejected target overrides the positive response of a nearby, normally accepted one, preventing perception of the good item. This problem can be reduced by avoiding excessive sensitivity and discrimination, selecting a narrow-scan or small diameter searchcoil, and scanning the area from several different directions.

TEN-TURN — A type of control used for fine tuning of the threshhold of ground balance. The control manually rotates ten turns to cover the full electrical range of the function before reaching the end of its range.

TEST PLOT — A mapped area (sometimes called a "test garden") in which targets are buried at various depths and allowed to remain in the ground for a time, simulating actual field conditions of moisture, oxidation, etc. Use of a test plot can aid in developing operator skills and evaluating new equipment or methods under known ground/target conditions.

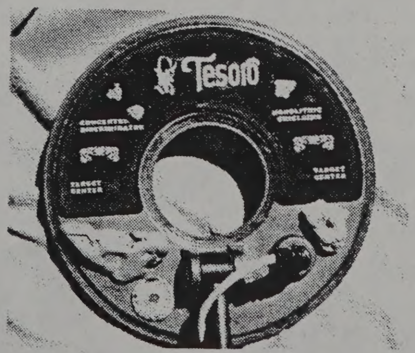
THRESHOLD — The optimum tuning setting for a metal detector, at which a faint buzz or hum can be heard — the threshold tone. Note: This concept does not apply to "silent search" operation.

TR — Transmitter-Receiver. A type of metal detector circuit in which a transmit oscillator generates a specific rate of current frequency through a multiple-wound searchcoil, creating an electromagnetic field. Detection is achieved by target surface eddy current generation and inductive imbalance between the transmit and receive windings. TR is often used to designate early low-frequency (LF) detectors of this type, as opposed to more recent very low frequency (VLF or VLF/TR) designs. It is also used to signify the non-motion discriminate mode.

VLF — Very Low Frequency. A radio frequency spectrum ranging from 3 kHz to 30 kHz, and the one most commonly used in current metal detector designs. VLF is also used to designate TR detectors operating at very low frequency — VLF/TR's — and is sometimes substituted for the mode of operation known as all metal or normal.

VLF/TR — A class of TR metal detectors operating in the VLF range. Also used to designate detectors of this class which do not include a motion-discriminate mode; in such cases, a distinction is made between VLF/TR's and "motion discriminators," even though the latter instruments also operate on the TR principle and in the VLF range.

VOLUME COMPARISON — A technique used to determine approximate target depth when operating a non-motion VLF/TR metal detector. After receiving a signal while searching in the discriminate mode (adjusted to a low rejection level), the operator switches to the all-metal mode and re-checks the signal. If the signal intensity is approximately the same in both modes, the target can be considered possibly deep and worth investigation. If the signal is much stronger in the all-metal mode, the target can be considered possibly shallow.



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Salisbury
Report

Placer Mining Technology and Associated Environmental Effects

Prepared for
The National Park Service
Energy, Mining and Minerals Division
P.O. Box 25287 • Denver, CO 80225 • 303-236-8777

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PLACER MINING TECHNOLOGY
and
ASSOCIATED ENVIRONMENTAL EFFECTS

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February, 1987

Prepared for the National Park Service
Energy, Mining and Minerals Division

TABLE OF CONTENTS

CHAPTER 1

PLACER GEOLOGY, EVALUATION AND MINING METHODS

CHAPTER 2

ENVIRONMENTAL IMPACTS

CHAPTER 3

WATER QUALITY

CHAPTER 4

RECLAMATION

LIST OF FIGURES

PREFACE

This handbook was prepared by Salisbury & Assoc., Inc. (SAI) of Spokane, Washington, under contract to the National Park Service (NPS). Chapter 1, entitled Placer Geology, Evaluation and Mining Methods, was written by Jeffrey H. Levell. Chapters 2,3 and 4, respectively entitled Environmental Impacts, Water Quality and Reclamation, were written by Larry A. Peterson and Gary E. Nichols of L.A. Peterson & Associates of Fairbanks, Alaska, under subcontract to SAI. Editing, editorial additions and the section on water-related regulatory requirements at the end of Chapter 3 were provided by NPS personnel. The Introduction to this handbook and the Introduction and Summary to Chapter 2 were written by Sidney L. Covington with the NPS, Energy, Mining and Minerals Division in Denver, Colorado.

INTRODUCTION

PURPOSE

This publication is intended to be a ready-reference handbook and guide for park and regional staff who will be called upon to evaluate the technological aspects of proposed placer mining operations and the environmental impacts of those operations. Those operations may be within a park unit, and therefore covered under a plan of operations; or, they may occur outside the park boundary in such proximity as to effect the resources within the park. This handbook should be an aid in the understanding of the equipment and methods used in placer mining, provide a background to assess environmental effects of placer mining, provide potential mitigation measures for environmental impacts and give a brief synopsis of the current environmental regulations pertaining to placer mining.

The text is not intended to be exhaustive, but rather to provide a synopsis of the placer mining industry and at the same time provide references which can lead the reader to a more in-depth study, if desired. In addition, it is intended to emphasize those aspects which are important to the evaluation of a placer mining operation from a National Park Service (NPS) perspective. The user should call upon expert help for additional information, including NPS staff geologists and engineers.

Since the great majority of placer mining affecting units of the National Park System occurs in Alaska, most of the discussion is written with Alaska operations in mind. Discussions of fish, wildlife and specific environments are for Alaska. However, most of the mining and sampling techniques may be applied to placer operations universally.

BACKGROUND

In 1976 the U.S. Congress enacted the Mining in the Parks Act (16 U.S.C. 1901 et seq.) closing all units of the National Park System to the location of new mining claims and directed that all mineral development activity in all units of the System be regulated. The NPS has been charged with this authority. NPS minerals management regulations are contained in the Code of Federal Regulations (CFR) Title 36, Part 9, Subpart A. These regulations were promulgated to control the conduct of mining and mineral related operations so as to minimize damage to the values and purposes for which the parks were created. The principal method that the NPS uses to enforce these regulations is to require plans of operations for all mineral exploration and development activities within NPS units and appropriate bonds for the performance of the mining and for reclamation. Discussions of plans of operations, bonding and applicable mining laws and regulations are beyond the scope of this handbook but may be found in other NPS publications. A discussion of water quality regulations is contained in Chapter 3 of this document.

When placer operations are proposed or conducted outside but adjacent to an NPS unit, it may be necessary for the NPS to work with the appropriate federal or state land management and regulatory agencies. On federal land these include the Bureau of Land Management, the U.S. Forest Service and the Fish and Wildlife Service. In Alaska these may include the Alaska Department of Natural Resources, Department of Environmental Conservation and Department of Fish and Game. It is important for the NPS to be intelligently involved in evaluating these proposed or active operations which will impact the park from outside its boundaries.

OVERVIEW OF CONTENTS

This handbook has been divided into four chapters with a glossary in the back. At the end of each chapter is a list of references applicable to the subject matter of the chapter.

Chapter 1,"Placer Geology, Evaluation and Mining Methods," describes what placer deposits are, how they are formed and where they occur. A section on placer evaluation methods is included to give the user an idea of the difficulty and problems in evaluating a placer deposit, and what an NPS mineral examiner would consider in examining a placer claim for validity. The last section in this chapter reviews gold processing and recovery techniques, historical and current mining methods and mining in three of Alaska's national parks.

Chapter 2, "Environmental Impacts," is written principally for the Alaska environment. Both the aquatic and terrestrial habitats discussed are those in Alaska. Nevertheless the principles which are discussed and the impacts to the environment may be applied to most any stream placer deposit. The last section in this chapter is a discussion of the actual mining activity as it involves impacts from the earth moving techniques and equipment.

Chapter 3 deals entirely with water quality and should be used with other publications, such as those listed at the end of the chapter, to give a more complete picture of water quality parameters and sampling techniques.

Chapter 4, "Reclamation" describes how both aquatic and terrestrial land reclamation may be accomplished. It should be kept in mind that land reclamation is not just the final step or aftermath upon completion of mining, but rather part of the mining process. Reclamation is often accomplished as mining proceeds. Mining plans, as a rule, should include an integrated reclamation plan.

In conclusion, this handbook is meant to provide quick information to NPS personnel in the field and to provide references and sources for furthur research if needed. It is not intended to be exhaustive. It should, however, be useful to the field person reviewing a plan of operations for placer mining and for monitoring a placer operation onsite.

CHAPTER 1

PLACER GEOLOGY, EVALUATION AND MINING METHODS

PLACER GEOLOGY

DEFINITION

In the discussion that follows, the term "placer" or "placer deposit" refers to deposits of unconsolidated sand, gravel and other detrital or residual material containing a relative concentration of gold or other valuable minerals. The valuable minerals have accumulated through natural processes of weathering, erosion, sedimentation and mechanical concentration. Placer deposit formation requires the following:

1. A valuable mineral which is relatively heavy and is resistant to mechanical abrasion and chemical solution. Some heavy minerals such as pyrite or cinnabar (mercury) may be found in placer concentrates but they generally do not travel far from their source because they are relatively brittle, soft or chemically unstable. Valuable minerals usually mined from placers are gold, silver, platinum, cassiterite (tin) and gemstones. Magnetite (iron), ilmenite (titanium), rutile (titanium) and monazite (rare earths, thorium) are mined to a lesser degree (Wolff 1980). The last four minerals are also common associates of gold in placers and are usually present in the placer concentrate after mining.

2. The valuable mineral must be released from its parent rock by natural processes of chemical and mechanical weathering.

3. Concentration of the valuable mineral. Concentration usually involves water transport.

"Placer gravel" is the rock waste or detrital material which makes up a placer deposit. Placer gravels may include colluvium, stream gravels or beach sand - each containing variable amounts of different sized material ranging from clay-to boulder-size.

This manual discusses placer gold deposits because placer mining operations in the United States and particularly in Alaska are predominately gold recovery operations. However, the principles of deposition, formation, evaluation and mining are essentially the same for all heavy mineral placer deposits.

FORMATION

The location, size and shape of a placer gravel deposit are functions of the regional forces of erosion, transportation and deposition which created it. However, its final form will be controlled by local conditions (Wells 1969). Each placer deposit will be absolutely unique in one or more ways - even within the same mining district or park area.

Placer deposits are produced primarily by weathering and disintegration of lodes or source rocks containing gold. According to Wells (1969), in most cases, the grade and size of a placer does not depend on the richness of

the primary source rocks, but rather, on an abundant supply of gold-bearing (auriferous) source materials and on physiographic conditions favorable for gold concentration.

Gold found in a placer deposit may be derived from a number of different sources, including the following:

1. Lodes, veins or mineralized zones - placers are commonly found in lode mining districts, but highly productive placer districts also occur in areas without obvious lode deposits. Preexisting lode deposits may have been destroyed by erosion or gold may be contributed by small mineralized zones. These small zones may be economically insignificant by themselves, but are abundantly distributed throughout a placer district.

2. Auriferous (even slightly) sulfide or polymetallic deposits or large porphyry deposits containing small disseminated amounts of gold. Numerous gold placers are located in lode mining districts which are not primarily gold districts.

3. Regional bedrock - occasionally formations of conglomerates, quartzite or other rocks will contain small amounts of disseminated gold. Also, formations of schists, gneisses or other rocks may contain small disseminated amounts of gold in slightly auriferous minerals such as pyrite or other sulfides.

4. Old placer deposits - older generations of placer deposits are often subjected to new erosion and alluvial transport by streams creating new and sometimes richer placer concentrations.

5. Glacial deposits - unsorted glacial deposits may contain gold from the gouging out of lodes or previous placer deposits. The action of recent streams may, in some cases, reconcentrate the gold after the glacier has retreated.

Principal mechanical agencies that assist in the transport and concentration of gold into placers are gravity, running water in streams and the agitation of waves along ocean shores. Of these, the action of streams is the most important in the movement and deposition of sediments and consequently in the formation of most placer deposits. Fluvial processes, alluvial material movement, and the mechanics of placer formation are beyond the scope of this manual. However, the following basic concepts of stream processes, summarized from Wells (1969), are necessary to the understanding of different placer deposit types encountered at mining operations in the field.

Erosion of the streambed occurs when the water flow has the ability to transport more material than is being supplied. Conversely, when the flow velocity is slowed or the water is overloaded with sediment, deposition of the sediments will occur. Where velocity is slowed locally, such as on the downstream side of boulders, other stream obstructions or on the inside bank of a stream bend, temporary deposition will occur. Streams generally transport most of their sediment load during periods of flood, and most deposition will occur when flooding stops. Coarse sand and gravel are intermittently bounced, pushed and rolled along the streambed. Much of the

coarsest cobble and boulder material is usually static except for brief periods of high-velocity stream flow.

Downward movement and concentration of gold and other heavy minerals on or near bedrock occurs when stream conditions are such that the entire bedload is shifted or sufficiently agitated. This movement and agitation of the alluvial material allows gold and other heavy minerals to sink quickly to bedrock, concentrate in natural sedimentary traps and form typical stream placer deposits. Fine gold particles, however, may be dispersed throughout the total depth of the placer gravel.

Coarse, unsorted material is usually present in the upper reaches of a drainage system. During flood conditions relatively small gold particles will be swept up and deposited farther downstream, where coarser gold will settle more readily to bedrock. Once the coarse gold has settled, it is very difficult to dislodge. Thus coarse gold will generally be found in placer deposits close to the source rocks and finer gold deposits are located farther downstream.

Physical characteristics of the bedrock are important factors in the formation of natural sediment traps and consequently placer deposit formation. Cracked bedrock, with an uneven or hackly texture, such as slates, schists and other naturally jointed rocks are favorable, particularly if the structures are steeply dipping. These rocks create abundant riffle-like traps for capturing gold and protect it from high-velocity currents once it has been deposited. Clays or decomposed, clayey rocks are also effective in trapping gold particles. Gold particles often settle into cracks and crevices or clayey rock creating economic concentrations several feet into the bedrock. "False bedrock," usually in the form of a well-defined stratum of clay, compact sand or limonite-cemented gravel situated above the actual bedrock, may be present within some placer gravel deposits.

Higher bedrock concentrations or "paystreaks" often occur within gravel deposits. They may follow a sinuous course which has no obvious relationship to the present course of the stream channel, but reflect the location of former, abandoned channels.

DEPOSIT TYPES

Although the following discussion covers deposit types in Alaska, the types listed here are generally the most common placer deposits located in the rest of the United States as well.

Eluvial (Residual) Placers

The first stage in the concentration of gold takes place at or near its release from the source rock. Weathered parent rock undergoes chemical, mechanical and biological weathering resulting in the loss of valueless material and relative enrichment in gold by leaching and winnowing. These deposits can be relatively rich but are usually restricted in size. According to Wolff (1980), eluvial placers are uncommon in Alaska because weathering conditions are unfavorable for their formation. Those that occur

probably owe their enrichment to mechanical breakdown of enclosing rock, creating a naturally broken and easily minable deposit, rather than to the removal of valueless matrix material.

Colluvial Placers

These deposits are located between eluvial and stream placers. They are usually in the form of surface detritus and soil and are limited in extent. Surface creep and a number of other erosional transport mechanisms, such as rain splash, sheet flow, solifluction, mud flows and land slides, move the rock waste and gold down hill. Relatively lighter material may be winnowed out of the deposit by rain wash and wind, and as the whole mass gravitates downhill, a rough concentration of gold may develop. According to Wolff (1980), this type of placer is not abundant in Alaska and they are usually low grade if present. Their chief economic value is that they move the gold into drainage systems to be further concentrated.

Stream Placers

This type of deposit is frequently the subject of mining operations in the western United States and Alaska. They are found in drainage systems ranging from steep gradient gulches in the upper reaches to lower reaches characterized by flatter stream gradients with wider and often extensive beds of gravel. Each individual deposit is different from the next because of unique local conditions of erosion and deposition. Individual deposits are characteristically different in the manner in which the gold is deposited and consequently in the grade of the deposit and the method in which they can be mined economically (e.g., size and type of equipment used). Following are descriptions of general types of stream deposits currently mined:

Gulch Deposits - Located where stream flow commences in the source area. Gulches are characterized by steeply incised canyons, steep gradients, and contain narrow deposits which are generally small in areal extent (Figure 1). Rapids and waterfalls are common and stream flow may only take place during periods of high precipitation or thawing. Gravel accumulations are often thin, discontinuous and poorly sorted with abundant boulders. Gold is commonly coarse with a large percentage of nuggets. Gold values are usually sporadically distributed but well concentrated on and in bedrock. These deposits can usually be mined only by simple hand mining techniques or with very small mechanized equipment.

Creek Deposits - Creek deposits are very similar in form to gulch deposits, but physical characteristics are less extreme, in that they have flatter gradients and wider deposits (Figure 2). They are important sources of gold in many Alaskan districts. Larger deposits and more space accommodate larger mechanical equipment and consequently larger mining operations. Most are mined utilizing some form of sluice or other recovery system and various combinations of compatible mechanical excavating equipment.

River Deposits - Several gulches and creeks eroding a drainage basin will join together at lower elevations to form larger rivers and deposits characterized by more extensive gravel flats (often several hundred feet

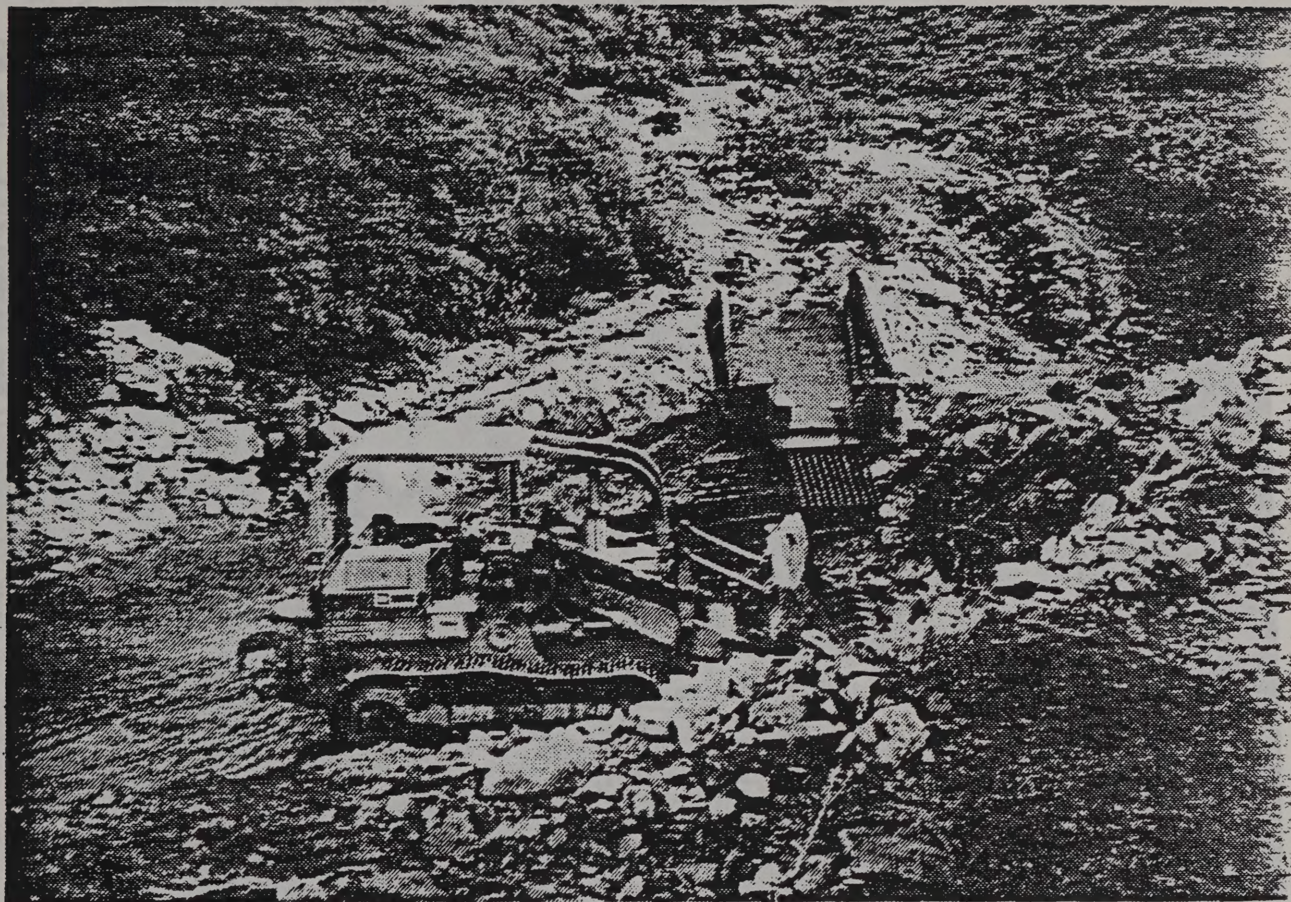

Figure 1.- Gulch deposit illustrating small working space and coarse, shallow gravel deposit.

Figure 2.- Creek deposit at the confluence of two gulches.

across) (Figure 3). Because of the shallower gradient, the carrying capacity is limited, and consequently a more uniform gravel size distribution is present along with a more uniform dissemination of gold. Gold deposited is usually finer-grained, more worn, and flatter than that found in creek deposits, and the overall deposit may often be lower grade. However, paystreaks and bedrock concentrations are often capable of supporting large-scale mining operations and occasionally dredging (Wells 1969).

Gravel Plain Deposits - These deposits are commonly found in the floodplain of large rivers where the river flattens and widens. Vertical erosion is less rapid and the stream begins to move laterally, creating meanders, shifting stream channels, and braided channel systems. Major deposits often occur where abrupt changes in the gradient occur, such as at the confluence of two streams or where the river breaks out from an area of hills into larger valleys or plains. Gold deposits are comparatively richer here because the stream-flow velocity lessens and loses its transporting power. Gold deposited in this lower area of a drainage system will be fine-grained, principally because of the lower water velocities available to carry it this far downstream. Although part of the contained gold will be concentrated at or near bedrock, much of it may be fairly uniformly distributed vertically and laterally throughout the gravel deposits because of the constant lateral migration of stream channels. These deposits are usually low grade and, if economic, are mined using very large-scale mining and dredging equipment. Because of the small size and flatness of the gold it is difficult to recover and requires relatively sophisticated recovery techniques, such as jigs.

Bench (Terrace) Placers

These deposits are remnants of alluvial valley fill formed during an earlier cycle of erosion and deposition. They are left stranded along one or both valley sides when the stream incises its bed downward in response to a relative lowering of the base level (see Figure 3). The abandoned gravel deposits usually follow the course of the present day stream and approximate its gradient. More than one level of bench deposits frequently occur. Benches are often mined in Alaska (and elsewhere), and usually require large-scale, mechanized operations. Bench gravels in Alaska are often frozen and require thawing prior to mining.

Beach Placers

These deposits form where gold-bearing material is supplied to the ocean shore by streams or possibly downhill creep along the shoreline. The deposits are formed by the action of waves, tides, undertow and longshore currents. If the position of the shoreline remains fairly constant, gold may be concentrated by continual wave agitation and the washing away of the lighter material. These placers are most often long and narrow and usually contain very fine and thin, flaky gold which is difficult to recover by conventional gravity methods. Some of the best examples of beach placers are those at Nome, Alaska, on the Seward Peninsula, where several ancient, elevated deposits are located inland from the present coastline.

Figure 3.- River and parallel bench deposits. Rectangular cut in central foreground is from recent mining of frozen gravels on the bench.

PLACER EVALUATION

Exploration for placers in the past was usually a very simple process of prospecting stream gravels to find the best accumulations of gold. Sampling was carried out by hand methods of pitting or sampling existing exposures of gravel. Samples were washed by panning or crude sluicing. In the early days, and to some extent even today, if good values were obtained from a few tests, the prospector immediately turned miner and the richest paystreak material was mined. If returns were poor, the prospector continued on.

Easily accessible and high-grade deposits are rare today, and lower grade deposits require the use of more costly, mechanized machinery and recovery systems in order to mine the deposits efficiently and economically. Some form of systematic exploration is therefore required to provide reliable ore reserve estimates and reduce risk. Evaluation of a placer deposit is only the initial phase of a complete mining operation.

EVALUATION GOALS

The goal of any evaluation program is to define the size and grade of a placer deposit and to determine whether it can be mined at a profit. Initially, an evaluation program may include broad-scale geologic studies using a large array of technological aids to locate new deposits or previously unknown deposits in a drainage system with a history of placer mining. Once an area has been explored and a prospective deposit has been identified, more detailed studies are required to determine the parameters which directly affect proposed mining, rehabilitation, gravel processing and gold recovery techniques. The goal of the evaluation is to determine the volume, grade and geometry of the deposit as well as gravel and overburden characteristics, distribution of gold values within the deposit, configuration and type of bedrock, conditions affecting the environment and, particularly in Alaska, amount and distribution of permafrost.

In order to plan a successful mining operation, a good sampling program is the most critical and valuable source of information and will provide much of the data on the physical conditions previously described. Although not detailed in this manual, several other pieces of information should be gathered during evaluations, all of which affect the commercial value of a deposit. These include deposit access, local topography and water availability.

EVALUATION PROBLEMS

Representative placer samples are desirable but usually difficult to obtain for the following reasons:

1. Obtaining a representative sample requires sampling the nonhomogenous mixture of clay, sand, gravel and boulders in their natural proportions in the sample (Figure 4). In addition, these proportions commonly change - often radically - over very short distances, both vertically and laterally throughout the deposit.

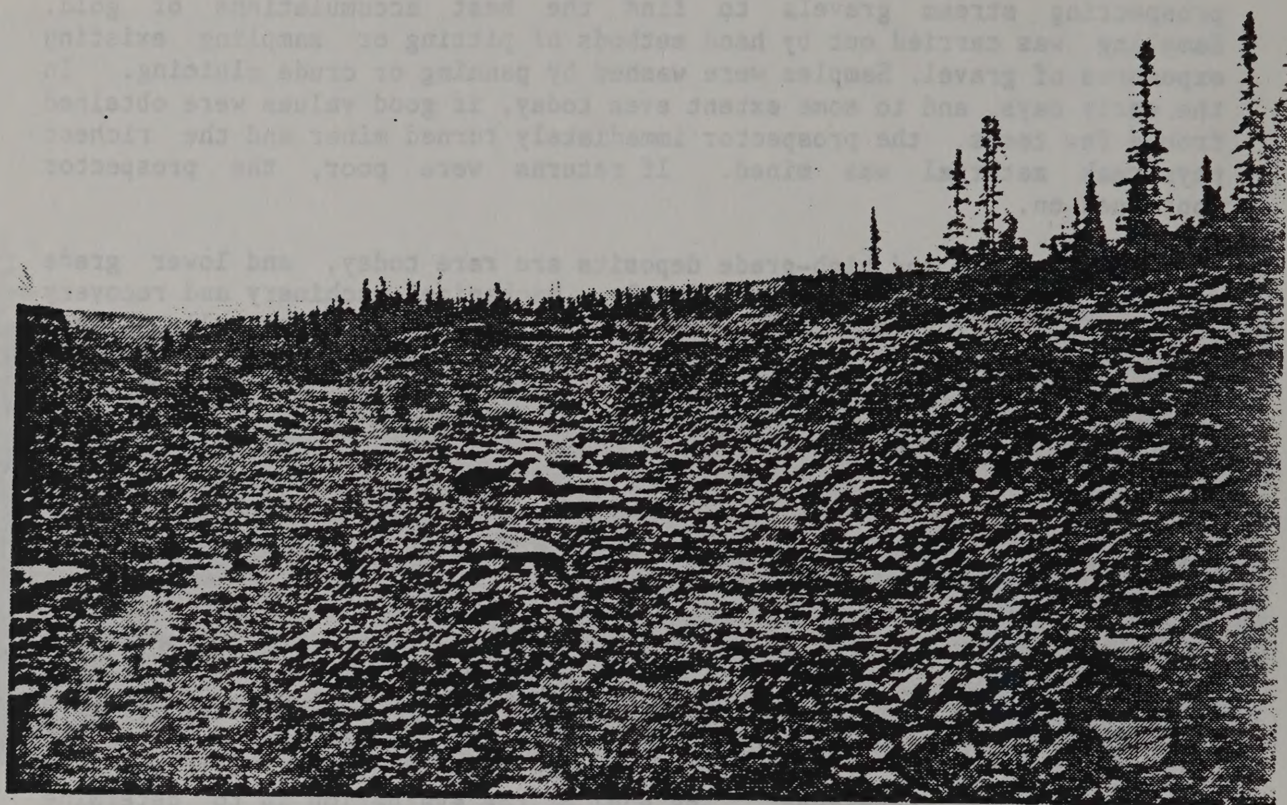

Figure 4.- Typical placer material in a bench deposit illustrating a wide range of particle sizes.

2. In most placers, gold is not uniformly distributed throughout the whole mass and is often quite erratically distributed. Gold may also be confined to narrow, sinuous paystreaks. If gold is relatively evenly distributed within a paystreak, the paystreak itself is difficult to locate within the other low-value gravels. Also, coarse gold deposits are even more erratic and values may be concentrated in zones tens to hundreds of feet apart.

Collecting large samples to offset the possibility of including or excluding high-value gold particles (nugget effect) is often impractical and may be tantamount to actual mining of the property. Another alternative is to take an abundant number of samples to statistically increase the degree of confidence in the results. Economic and time considerations often render this solution impractical also.

3. When dealing with gold placers, the high unit-value of gold causes a problem. According to Wells (1969), in a commercial placer the relative amount of gold (by volume) may be on the order of one part gold to a hundred-million parts of gravel. As a result, any error in gold content of the sample will be highly magnified in the end calculation of grade.

From large-scale, company-sponsored programs to the individual prospector, the evaluator must be aware of these problems when planning a sampling program and evaluating sample results.

EVALUATION METHODS

Placer evaluations are usually conducted in two phases, the first being regional or large-scale exploration reconnaissance studies to locate a potentially economic deposit. The second phase involves sampling and gathering detailed data on a specific deposit in order to delineate enough commercial gold-bearing gravel (paydirt) to justify development and mining or to eliminate the deposit from further consideration.

In this manual, it is assumed that the first phase of exploration has been conducted and any or all of the following favorable geologic conditions have indicated a high probability that minable deposits may exist and a specific deposit has been targeted for further evaluation:

1. The deposit is located in an area which drains mineralized, known, gold-bearing terrain.

2. Reconnaissance sampling indicates placer gold is present.

3. The drainage or neighboring drainages have a history of placer mining.

The following discussion deals with the evaluation of a specific deposit.

When planning a sampling scheme, consideration should be given to the physical characteristics of the deposit, such as probable source of the gold, distance the gold has traveled and erosional forces that contributed to formation of the deposit. Generally, the larger and more uniform the gravel deposit, the more evenly distributed the gold values will be. In this case, sampling data may be projected up to several hundred feet. In comparison, it may be risky to assume an area of influence of over a few

feet from a sample obtained from a coarse-gold gulch deposit. Conversely, negative values from a coarse-gold deposit may not necessarily indicate minable reserves are not present.

A few samples will provide valuable information about a deposit, but they are usually not sufficient to accurately determine the overall grade. A detailed sampling program is usually required to reliably estimate the true potential of a deposit. However, lack of time and exploration capital and need for cash flow require some placer mine operators to abandon the detailed exploration approach and start mining. Their approach may be somewhat unscientific but based on their experience and empirical knowledge, they may be satisfied with the degree of certainty obtained from a few judiciously chosen samples. Samples may be collected by simple techniques of hand panning existing exposures or perhaps by backhoe trenching.

More sophisticated programs may similarly include a few preliminary "hand" samples to determine if a deposit has potential, but this work will be followed by systematic trenching or drilling. Final evaluation may include a bulk sample or scaled down pilot mining program using equipment similar to that contemplated for the final mining operation. Common methods of sampling are discussed in the following paragraphs.

Existing Exposures

Sampling stream banks, road cuts or old mine pit walls can give the investigator immediate results from a small investment of time and money. Although bedrock material may not be included in the sample and sampling locations may be unevenly distributed, this form of sampling can provide useful preliminary data. A few small samples can help determine if gold is present and provide a rough estimate of the values that may be expected. Samples are usually concentrated by hand panning or with a small sluicing device using simple gravity separation.

Trenching

Trenching may be done with either a backhoe or a bulldozer. In relatively shallow deposits, trenching with a backhoe is an effective and versatile sampling technique (Figure 5). Depending on the size of the machine, depths of up to 30 feet are attainable. It can be used to dig pits in selected locations in a reconnaissance program or as a detailed investigative tool to dig pits in a systematic grid pattern. Another valuable application is its ability to trench across the entire width of a stream drainage or parallel to the drainage as an aid in defining other physical characteristics of the deposit.

The evaluator can either bulk sample the entire excavation or cut channel samples in the walls of the trench. Advantages of this form of sampling are the evaluator can: 1. observe physical characteristics of the alluvial material, 2. obtain a large and complete sample often including bedrock and 3. if the trench remains open, take check samples at a later date.

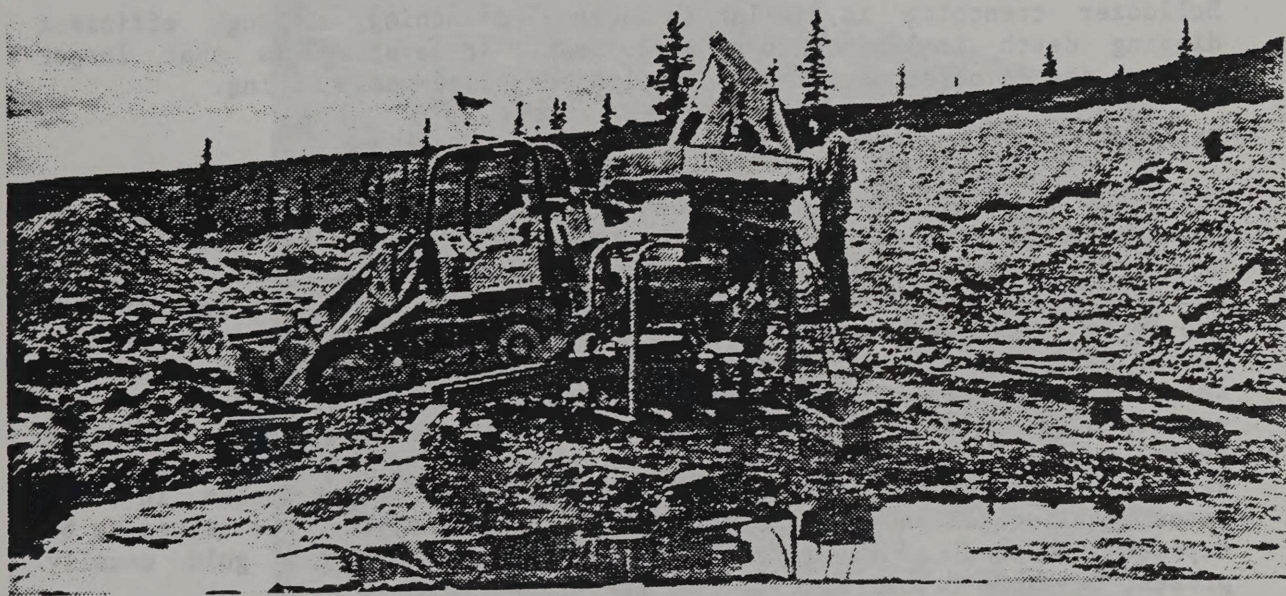

Figure 5.- Backhoe trenching using a Denver Gold Saver to process and concentrate the bulk sample.

These large-volume samples can be washed with small sluices, rockers or with small mechanical washing plants, such as the Denver Gold Saver (Figure 5), which uses the basic principles employed on most production machines. The typical machine consists of a screening device for sizing gravel and a short sluice for concentrating the gold. Sluicing is often accompanied by a shaking action. Small washing plants weigh a few hundred pounds and can be hauled by trailer, truck or backhoe. Washing plants can be quickly and easily cleaned eliminating the danger of contamination from one sample to the next.

Bulldozer trenching is similar to backhoe trenching although efficient digging depth is limited to about 10 feet. An advantage is that larger samples can be obtained for bulk sampling or pilot-scale mining.

Drilling

Drilling is widely used for more detailed studies of placer deposits and especially in deep, wet or frozen ground where other methods are not practical. Drill samples are relatively small and can be less representative because of their increased sensitivity to the nugget effect. The probability that a small diameter drill sample contains a true average number of gold particles is less than for larger diameter sample. However, the ability to sample deep and frozen deposits with relative speed usually outweighs this disadvantage. In addition, many economic placers currently being explored and mined are within larger river, gravel plain and beach deposits which contain fine gold in relatively uniform paystreaks. In these cases, small drill samples are more representative of the true values to be expected because of the more uniform distribution of fine gold usually present in these deposits.

Drilling programs are usually employed only by larger companies because of their high initial cost both in equipment and transportion. This investment is quite often greater than the independent prospector/miner can afford. Drills can be mounted on a truck or track vehicle or pulled by bulldozers. An advantage of drilling is that it can be conducted during the winter months and used to delineate reserves for summer mining. Recent technological advances have created a variety of fast penetrating drills for placer evaluation. Following are the most widely used drilling techniques.

Churn Drilling - This is the oldest and most widely used and accepted method of drilling placers (Figure 6). Percussion is used to drive casing through the alluvium and the sample is then chopped up (churned) using a chisel-shaped bit suspended from a cable. In frozen gravel, casing is generally not required except for a few feet at the surface to keep water and debris out of the hole. After a run of a few feet, a sand pump is lowered into the hole to recover the sample. The sample is measured to determine volume, concentrated (usually with small sluice or rocker) and then hand panned to further reduce the sample volume. Compared to other drilling techniques churn drilling is relatively slow, but it is still considered the most accurate method of recovering samples by most experts.

Rotary Drilling - The rotary drill is commonly used to drill blastholes in hard rock mining but it has also been applied to placer exploration. Drill

Figure 6.- Nodwell-mounted churn drill sampling frozen bench deposit. Evaluator in left foreground is concentrating samples by hand panning.

cuttings are forced out of the hole as the bit advances. Consequently sample volumes, accurate depth estimates and quantifiable grades are difficult to reliably assess. However, rotary sampling provides a relative idea of values and is particularly useful in providing other information on the physical characteristics of the deposit such as depth to bedrock. Another advantage of rotary drilling is its capacity to drill a variety of material at very fast penetration rates.

Reverse Circulation Drilling - Reverse circulation drilling is fast becoming an accepted form of obtaining a sample. This method employs a double walled pipe with an annulus between the inner and outer tubes. The annulus allows the introduction of air or water under pressure near the drill bit. Alluvial material penetrated at the bit is lifted upwards into the inner tube more or less continuously during the percussion driven penetration of the drill string. Penetration rates are far superior to churn drilling and very cost effective from a cost per foot standpoint.

Resonant Drilling - The resonant drill uses high-frequency axial vibration which produces a vertical stress component drilling action. In placer gravels, the transmitted vibration permits penetration by agitation and resorting particles at the bit face. Boulders and other competent material are penetrated by impact fracturing. Continuous rotation of the drill bit can also be superimposed on the vibrating motion to assist penetration. Samples are relatively undisturbed, and theoretically, all particles including boulders and gold, are preserved in the order in which they occur in the deposit. This characteristic is particularly advantageous because it permits detailed logging of the physical nature of the gravel and more accurate delineation of the auriferous gravel horizons. The core sample remains in the casing until the casing is pulled out of the hole. Penetration rates in sand and gravel can be as high as one foot per second.

Seismic Surveys

Depth to bedrock and bedrock configuration are important considerations when planning either sampling methods during exploration or in designing a mining method and selecting equipment. Seismic refraction or reflecton can be used to determine the location of bedrock under alluvial cover. This procedure involves bouncing sound energy off relatively more resistant bedrock to determine depth. However, seismic surveys may produce erratic or misleading results in frozen ground.

Magnetic Surveys

Because magnetite-bearing, black sand deposits are commonly associated with gold concentrations, magnetometers are occasionally used to delineate relative concentrations or paystreaks. However, concentrations of black sand are usually not great enough to distinguish between normal bedrock variations in magnetic response.

GOLD PROCESSING AND RECOVERY TECHNIQUES

Recovery of placer gold from a relatively large mass of gravel is primarily a gravity seperation process utilizing water as a transporting and seperating media and also utilizing the specific gravity differential between gold particles and the remainder of the alluvial material.

Gold Characteristics

Gold particles become rounded and almost always flattened to some extent by the processes of erosion and stream transport. Particle size may vary from large, coarse nuggets to minute "colors." The farther the gold has traveled from its source, the smaller and flatter the particles become. Smaller and flatter particles (fine gold) are easily buoyed up and carried long distances downstream, particularly in water containing much clay or talc. Although fine gold is present in most deposits, relative percentages will increase farther downstream in low-energy, flat-gradient drainages.

Fine Gold Recovery Problems

Gravity recovery methods are based on the physical properties of minerals such as size, shape and specific gravity and the resulting action in flowing water.

Size distribution of gold in placer deposits will vary widely. A study by Cook and Rao (1973) indicated Alaskan creek placer deposits contain from 15 to 25 percent of the gold as minus 100 mesh in size. In contrast, they showed gold recoveries, in this same size range from selected placer operations, were 0 to 5 percent. They further determined that it was extremely doubtful that conventional sluice recovery systems were very effective in capturing gold particles less than 65 mesh in size and that recovery of particles up to 20 mesh in size was impaired if the gold is flattened. Cook and Rao (1973) also estimated that up to 25 percent of gold may be lost to tailings, as fine or flat particles in conventional sluicing operations, with a greater percentage loss estimated for river bar deposits farther downstream. Although there is no standard definition of fine gold, the definition by Cook (1983) of minus 20 mesh to plus 100 mesh will be adopted here.

Although the specific gravity of gold is extremely high, particle shape and size have a significant effect on the relative settling velocities and consequently, the efficiency of gravity methods of recovery. Swift or clay-rich currents in a conventional sluice box can result in losses of fine, flat gold particles. The average particle weight of gold lost to tailings is lower than that in the heads because the gold particles in the tailings are flatter (Cook and Rao 1979). Classification (sizing) devices reduce the amount of water and associated higher velocities required to move larger gravel through a sluice box and increase the recovery of flat, fine gold particles.

Sluice Recovery Methods

Nearly all existing placer mining operations use a sluice for primary concentration and recovery. Transverse riffles are normally placed in the sluice box normal to water flow. Riffles are constructed out of a variety of materials. They may consist of wood, or more commonly, steel bars (often angle iron) or expanded metal. Riffles are often designed for easy removal and replacement to accommodate cleanups.

Riffles perform three functions: (1) retard the movement of gold flowing at the bed of the sluice, (2) act as a trap to retain the settled gold, and, (3) further classify material in the inter-riffle space by separating particles by differences in specific gravity (Cook 1983). Water flows through the sluice in laminar layers with higher velocities towards the surface. A rough surface (riffles) on the bed of the sluice creates obstructions which decrease flow velocity. In addition, the riffles promote turbulent flow which increases the ability to carry sediments in suspension. Turbulence increases rapidly with increased flow velocity. According to Cook (1983), where velocity is lowered on the downstream side of a riffle, higher density gold particles settle out. The eddying motion in the inter-riffle space keeps material in a continuous state of agitation. Fine-grained, lighter sediments are removed and lifted by hydrodynamic forces supplied by the turbulence and by impact from other particles. Coarser sand and rocks slide and roll along the bed as a result of mechanical drag and push of the turbulent water flow.

Successful gravity separation, where maximum gold recovery can be balanced with maximum gravel throughput, occurs when the feed, slope and width of the sluice and the amount of water used are at optimum proportions (Colp, 1979). Several variables affect the equation, such as the physical characteristics of the gravel and the gold or the quantity of water available or required to move the material through the sluice. No standard procedure exists to achieve an optimum recovery situation.

The grade or slope of the box is generally between 1 and 1.5 inches drop to 1 foot of box length (Colp 1979). This depends on the character of the feed gravel to be moved and the amount of water available to move it. Large, heavy material will require a steeper grade (increased velocity) and deeper flow of water and conversely smaller or preclassified and lighter feed material requires the opposite conditions. The sluice is preferably designed to accommodate slope adjustment.

Box width is also related to the volume of water, slope of the box and character of the gravel and gold. If the sluice is wide the water depth may be shallow and more conducive to fine gold recovery, but sanding may also occur. If relatively narrow, the depth of water and velocity will increase, facilitating transport of coarse material. Coarse gold will be recovered but fine gold may not have a chance to settle to the bottom.

The length of the box must be sufficient to thoroughly wash the gravels and allow the gold space to settle out. A majority of the gold will be recovered at the head of the box. The longer the box the more gold, and

particularly fine gold, will be recovered. This has practical limits as increasing the length decreases the ease of mobilizing and resetting the system, and increases cleanup time.

Constant feed rates are required. Surging of the feed causes the bottom of the sluice to alternately sand up or be washed clean reducing efficient gravity separation. Also, a steady and preferably adjustable water volume, "tuned in" to these parameters, will help maximize gold recovery.

The gravity separation process is more efficient when the feed is classified (sized). Density differences are magnified when the gold and other sediment particles are more closely sized. The sluice can subsequently be designed to accommodate moving finer-grained material and enhance fine gold recovery. Some classifiers are equipped to scalp off large boulders and gravel, and may also separate two or more size ranges. Different sized material can then be channeled into separate sluices designed specifically for that size range.

Undercurrents are an in-sluice sizing device. Undercurrents usually consist of a punchplate with small (generally about 3/8 inch) holes arranged in staggered rows. The punchplate is usually placed at the head of the sluice allowing fine material to drop below. Ideally, the fine material is channeled into another sluice specifically designed for recovering fine gold.

Fine gold recovery is also increased by placing plastic indoor-outdoor carpet or similar matting on the bed of the sluice. The fibers of the carpet create a safe lodging place for fine gold.

Specialized Recovery Devices

If a large percentage of the gold in a deposit is fine, it may be economical to use more efficient devices to improve its recovery. Several alternative devices have been developed to provide for fine gold recovery. Specialized recovery devices include jigs, pinched sluices, cones, spirals, bowl concentrators, cylindrical concentrators, tables, belt concentrators and cyclones. All the devices rely on gravity or centrifugal force to accomplish concentration and require a limited range of presized material to achieve optimum recovery. A few of these devices, especially jigs and spirals, are employed in opencut mines either as the primary concentrator or as an addition to the recovery circuit for processing classified feed from the sluice plant.

These devices require a significant capital investment for purchase, installation, operation and maintenance. To determine the need for acquiring a specialized recovery device, the operator must first know what amount or proportion of gold is being lost to the tailings. The economics of purchasing an improved device is then weighed against the monetary gain from improved recovery, or simply applying relatively inexpensive improvements to the sluice design.

Sluicing plants have a relatively low capital and operating cost and require only a moderate amount of relatively unsophisticated maintenance. In addition, sluices can also be operated earlier and later in the mining

season, where the use of jigs and spirals is often curtailed (Peterson 1986). Without additional investment in heating devices, additional piping for ore slurries and water will become inoperable during freezing conditions in late spring and early fall.

According to Mildren (no date), specialized recovery devices can be roughly divided into three basic groups: 1. large volume, high capacity units where processing of large volumes are required, 2. medium capacity units for handling moderate volumes, and 3. low capacity devices for upgrading primary concentrates or handling low-volume, high-value material. Low capacity devices are often used in sluice concentrate cleanup and final gold separation in opencut mining operations. Often, some combination of the three types are utilized for efficient recovery. Operating principles of the most widely employed equipment are described below.

High Capacity Equipment

Jigs are the most commonly used device. The conventional jig is an open, water-filled tank with a screen near the top. The tank usually has several compartments with a spigot on the bottom for collecting and removing concentrates. Mechanical diaphragms or plungers create vertical pulsations in the water as the slurry is fed over a layer of coarse, heavy particles (often steel shot) held by the screen. This action takes advantage of settling velocity differences and causes a separation of heavy minerals which migrate downward while the lighter minerals are carried away by cross flow. It is one of the most efficient separators for material in the 1 inch to 100 mesh size range (Mildren, no date). Jigs can process from 1 to 75 cubic yards per hour, and are often used in series to create a progressively more concentrated product.

Other high capacity devices are the spiral conncentrator, the Reichert Cone, the pinched sluice and the Wright Impact Tray. Spirals and cones are most often used. In the spiral concentrator (Figure 7), the feed slurry is introduced into a vertical spiral channel where the minerals begin to settle and classify. The heaviest particles move to the inside path and are split off into collecting ports. Spirals can recover gold particles down to 325 mesh size. The few spirals used in Alaska, concentrate sluice tailings. The Reichert Cone (Figure 8) is based on the pinched sluice principle where the ore slurry is crowded into a progressively narrow opening. Gold migrates to the bottom forcing lighter minerals up. The slurry is discharged in a segregated flow and the heavy gold concentrate is split off.

Medium Capacity Equipment

Equipment in this group includes bowl concentrators (including centrifugal amalgamator), hydrocyclone, Bartles-Mozley table, Johnson-type cylinder, plane table and riffle table. Most of these devices were developed outside the United States, where placer deposits are being mined and processed for various other heavy minerals, such as rutile, ilmenite, zircon, cassiterite and diamonds (Mildren, no date). Although this group of equipment is less appropriate for high-volume placer operations, according to Mildren (no

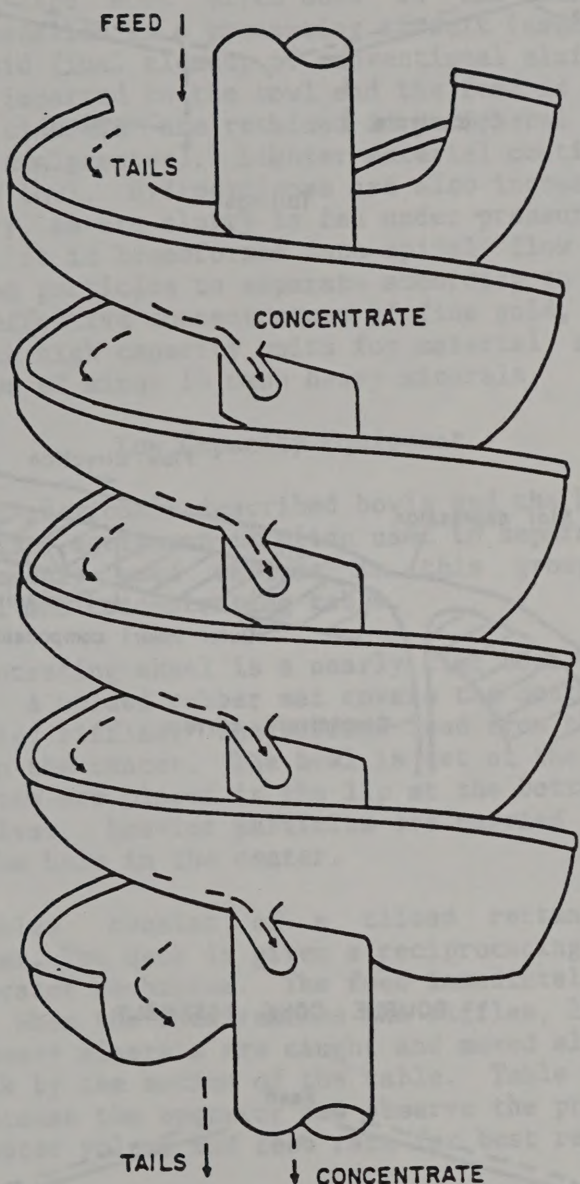

Figure 7.- Spiral concentrator.

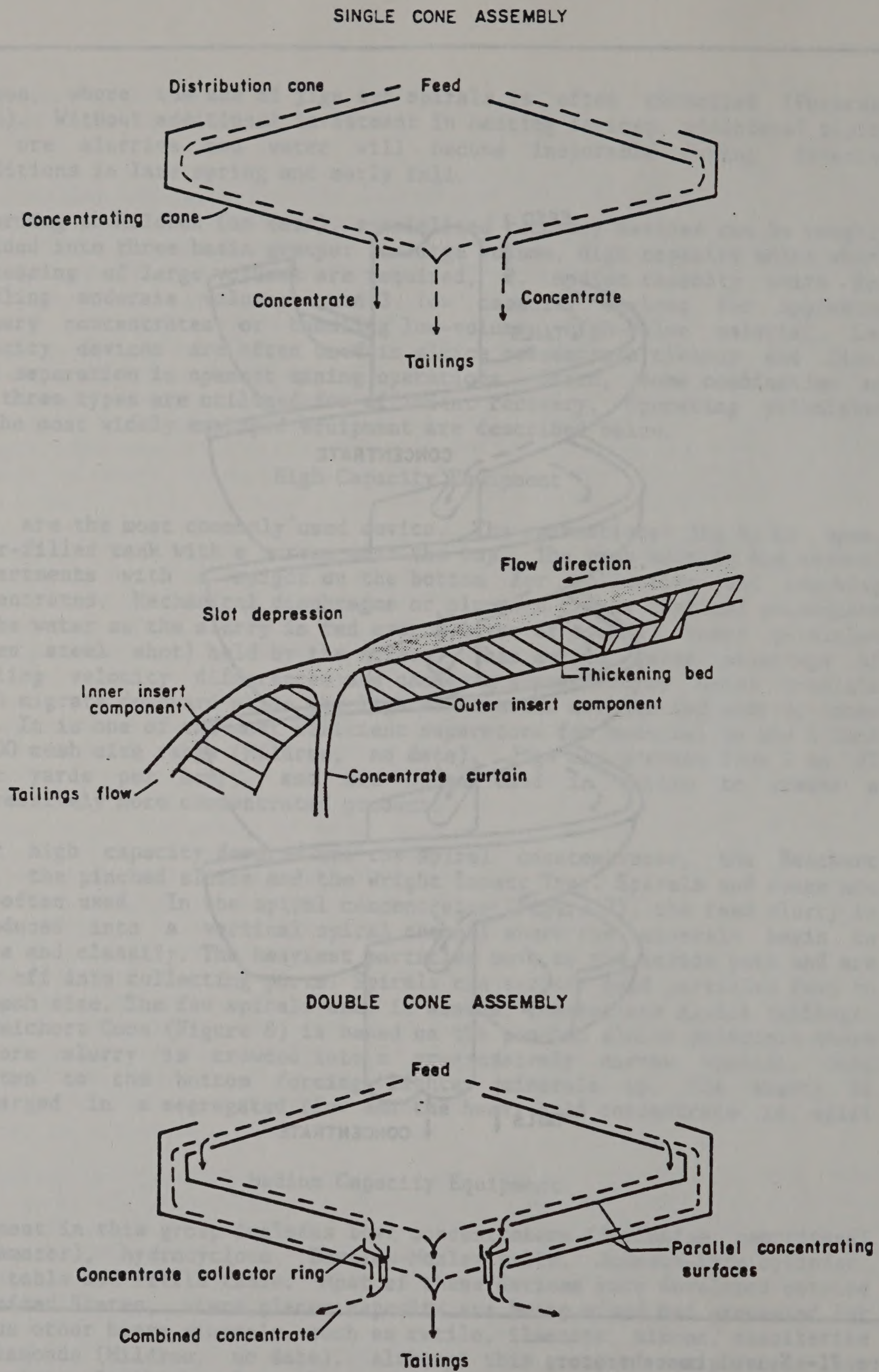

Figure 8.- Reichert cone concentrator.

date), processing of moderate volumes may be economically feasible in relatively rich deposits or in isolated areas where transport of equipment or scarcity of water are problems.

Bowl concentrators are most often used in the United States placer industry, either installed in a processing circuit (usually to treat sluice effluents), or to aid final cleanup of conventional sluice concentrates. A spinning motion is imparted to the bowl and the feed is accelerated to the periphery. Heavy minerals are retained in peripheral riffles or in a recessed channel (amalgamator). Lighter material continues up and exits over the rim of the bowl. Hydrocyclones are also increasingly used by the placer gold industry. As ore slurry is fed under pressure into a stationary cone-shaped bowl, it is transformed into spiral flow where centrifugal acceleration causes particles to separate according to size and density. Although they are effective concentrators of fine gold, they are often used in conjunction with high capacity units for material size classification and preconcentration of minus 10 mesh heavy minerals.

Low Capacity Equipment

In addition to the previously described bowls and the Denver Gold Saver, other small capacity equipment is often used to separate gold from mine concentrates. Commonly used devices in this group are the spiral concentrating wheel and concentrating table.

The spiral concentrating wheel is a nearly flat bowl with an adjustable angle (Figure 9). A molded rubber mat covers the bottom of the bowl and consists of spiraled riffles. The riffles lead from the perimeter of the bowl to a hole in the center. The bowl is set at the desired pitch and presized concentrates are placed in the lip at the bottom edge of the bowl. As the bowl revolves, heavier particles are carried vertically up the riffles and into the hole in the center.

Concentrating tables consist of a tilted rectangular table with longitudinal riffles. The deck is given a reciprocating longitudinal action by means of a vibrator mechanism. The feed immediately fans out when it contacts the deck. When the feed reaches the riffles, lighter particles are washed over and heavy minerals are caught and moved along the riffles to the end of the deck by the motion of the table. Table tilt and water flow are adjustable. Because the operator can observe the process, he can adjust the table angle, water volume and feed rate for best results.

Testing For Fine Gold Loss

Although some authorities estimate sluice recovery from 60 to 85 percent, with most losses in the fine gold fraction, in some instances, fine gold may represent only a very small fraction of the gold in a deposit and may not warrant concern. Even if a significant portion of fine gold is present in a deposit, enough coarse gold may be recovered to make the operation economically viable. To analyze the economic benefits of improving a sluice system or using alternative recovery technology in order to increase fine gold recovery, the actual size distribution of gold particles contained in the deposit must be known. In the past, most analyses were conducted using field techniques comparable in efficiency to mine recovery systems. This

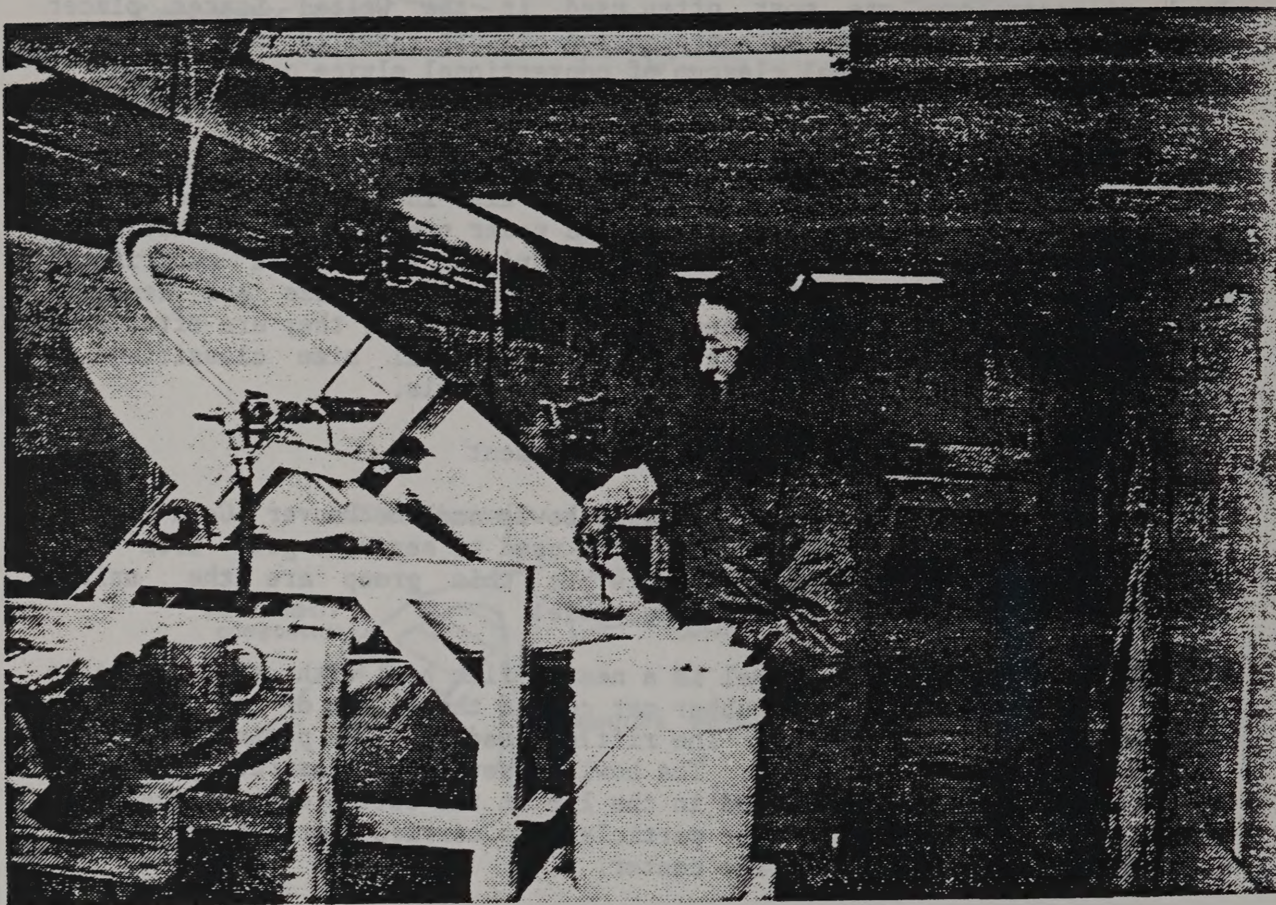

Figure 9.- Feeding a screened fraction of sluice concentrates into a spiral wheel concentrator.

did not give a realistic evaluation because of probable fine gold loss as a result of inefficiencies of the test recovery system. Some operators, however, conduct periodic sampling of the tailings in conjunction with size distribution analysis of gold recovered which will indicate how excessive losses may be. For a detailed study on testing techniques and recovery of fine gold, the reader should consult Cook and Rao (1973).

HISTORICAL REVIEW OF MINING METHODS

Placer mining was widely conducted in the western United States during the last half of the 19th century and in the early part of the 20th century. Placer mining by simple hand methods, using rockers and sluices, began in the 1870s in Alaska and progressively increased with the discovery of new districts and with the advent of larger-volume mining methods. By the turn of the century and up to World War I, ground sluicing, booming, hydraulicking and drifting in frozen ground had become popular mining methods. Conditions peculiar to Alaska such as, frozen deposits, adverse climate and a short mining season, allowed mining only the more easily accessible and richer deposits. After World War I, the less labor-intensive method of dredging had also found its application in Alaska. Dredging is discussed in a following section on current mining methods as it is still commonly used today and the basic principles and techniques are unchanged.

According to Wolff (1959), in the 1930's, development of diesel engine-powered bulldozers, draglines and portable centrifugal pumps provided improvements in gravel moving capabilities and further development of opencut/sluicing mining methods. In addition to less cumbersome methods of removing overburden or upper gravel layers, the bulldozer or dragline was commonly used to haul paydirt to the sluice or other recovery plant. The increased ripping power of the bulldozer also facilitated more efficient excavation of bedrock material. Mechanized equipment substantially increased the capacity to remove and stack tailings below the sluice. The use of this equipment also significantly reduced the large water requirement of earlier mining methods. Centrifugal pumps often eliminated costly ditch and reservoir construction and allowed recirculation of water in water scarce areas.

According to Wolff (1959), with the addition of versatile diesel equipment to the placer mining industry came the evolution of mobile, steel-constructed, skid or track-mounted washing and recovery plants. By World War II, a hopper set at the head of the sluice was often used which could receive feed gravels by dragline. Sluiceplates also came into use which allowed a bulldozer to push feed directly into the sluice. More easily adjustable water systems were developed using pumps, pipe and manifold spray bars to introduce water and initiate disintegration and slurrying of the paydirt. Immediately prior to World War II, Alaska was the leading gold producing state with an annual yield of nearly 750,000 ounces in 1940, most of which came from placer production (Peterson, et al. 1986). Gold mining was declared nonessential to the war effort, and nearly all placer mining stopped during World War II.

By the 1960's, inflation and the fixed price of gold had slowed the placer mining industry in Alaska to a near standstill. In 1971, because the price of gold began to increase, the industry began a significant resurgence and

newer mining methods employed backhoes, frontend loaders and scrapers in combination with bulldozers to haul and load paydirt and remove tailings. According to Peterson, et al. (1986), in the late 1970s, several hundred placer mines were operating and by the 1980s, over 500 were in production.

The advantages of material classification were generally recognized and systems from simple steel rail "grizzlys" to trommels and vibrator screens were added for rough classification by removing boulders and cobbles. Stacker conveyor belts were used on more plants to remove the oversize material from the classifier or to feed paydirt into the sluice or recovery plant. By the mid-1970s some operations were using settling ponds below the sluice, often for purposes of recirculating water and to help alleviate water quality problems.

The relatively recent use of mechanized machinery created much more efficient mining methods. This was necessary to mine increasingly lower grade deposits. In recent years, mobile, self-contained washing and recovery plants of all sizes have become common in many placer districts. These plants are mounted on skids, crawler tracks or tires. They usually contain some form of preclassifying equipment, a primary sluice, possibly other specialized gold recovery devices, and often conveyor stackers. According to Peterson, et al. (1986), much of the innovation in placer mining methods since the mid-1970s, and especially over the past 5 years, has been largely directed towards improving and modernizing the recovery system to more efficiently process and classify gravels, reduce water use and increase recovery of fine gold.

In recent years, many placer mining operations have adopted technological advances for water quality control. The addition of equipment for size classification and for fine gold recovery has resulted in a reduction in the amount of water used and consequently less water for treatment and removal of sediment content. Settling ponds constructed to treat mine operation effluent are currently required and many mines recirculate wash water - further reducing the amount of clear water used by as much as 50 percent.

Although seldom used today in commercial operations, pre-World War I methods of mining are described in the following paragraphs.

Rocker

Besides handpanning, this is the simplest method of washing and concentrating placer gravels or sluice concentrates. The rocker is relatively small and consists of a frame with rockers onto which a short sluice and a screened hopper or washing box are assembled. Gravel is shoveled into the hopper. The short sluice section, usually containing riffles, retains the gold. A rocker handle is attached to assist in the concentration process by manually rocking the whole unit back and forth while adding water with a ladle or small hose. This method is often used today to concentrate small drill samples.

Simple Sluice

This concentrating device has been widely used in most mining methods and washing plants up to the present day. In its simplest form, the sluice is a rectangular trough fitted with some form of transverse or lateral riffles. The riffles create an eddying and boiling effect causing the heavy gold particles to settle out and lodge in the protective cover of the riffle. In the past, burlap or canvas was laid under the riffles to assist in capturing the gold particles. The miner situated the sluice on the ground, introduced water for washing, adjusted the grade to most efficiently move the material through the box, and shoveled the paydirt into the box.

Ground Sluicing

This method is capable of processing much more material than simple sluicing but also requires considerably more water. A trench is excavated to bedrock along one side of the lower end of the deposit. A stream of water is then directed through the trench from a dam constructed upstream or by flume or pipe. The stream flow is directed to undercut the bank material. Paydirt may also be picked and shoveled into the stream. Natural depressions in the bedrock serve to concentrate and capture the gold. A sluice is commonly used to further concentrate the ground sluice concentrate. Another method also uses a sluice downstream to capture fine gold not trapped on bedrock.

Booming

This method is an alternative to ground sluicing employed where water is scarce. A dam equipped with a gate is constructed upstream. At certain intervals the gate is opened to allow a sudden rush of water to wash the gravel over bedrock. Often the gate is constructed to operate automatically whereby a certain depth of water in the impoundment will trigger its opening.

Hydraulicking

This is a relatively inexpensive method of working placer deposits and in the past, it was used in large-scale operations, processing up to several thousand cubic yards a day. A jet of water discharged under high pressure from a nozzle ("giant" or "monitor") excavates and "caves" the gravel banks by water impact. The gravel is transported by the water current to a sluice for concentration and recovery of the gold (Figure 10). Large amounts of water are required. The environmental damage created by hydraulicking has resulted in it being phased out or banned in most areas of the United States.

Drift Mining

Rich paystreaks in frozen deposits were mined by underground methods avoiding the need to handle valueless material or overburden. The paystreak was either accessed by sinking shafts and hoisting material from above or adits were used on some bench gravels to tram the paydirt. The paydirt was

Figure 10.- Hydraulic mining. Miner is using giant spray to wash feed gravels into the throat of the sluice. (Photo courtesy of Professor Donald J. Cook.)

sluiced in aboveground facilities to recover the gold. This method could be conducted during winter months but required thawing and rehandling the paydirt prior to processing in the summer.

CURRENT MINING METHODS

Placer mining is essentially an excercise in cyclical earthmoving involving: 1. excavation, transportation and stockpiling overburden and low-value gravels; 2. excavation and transportation of paydirt to the processing and recovery system; 3. processing the paydirt to recover the gold; 4. removal and storage of tailings, and 5. reclamation work consisting largely of backfill of tailings and stockpiled overburden and soils.

Opencut mining operations are different from one mine site to the next and can only be characterized on a site-specific basis. Although there are many similarities it can probably be stated that no two mines are identical. Several factors which are often interrelated dictate the mining, processing and recovery methods used at any particular mine site. These factors include the following: 1. deposit depth, width, overall volume and grade; 2. local topography and location of the deposit; 3. amount of water available for operations; 4. amount and type of overburden to be removed; 5. existence of frozen or thawed ground; 6. size and shape of gold particles; 7. excavating, processing, and recovery equipment available, and 8. environmental regulations and requirements concerning wastewater treatment, disposal of overburden and tailings, and reclamation.

The size and scale of mining operations varies with the size of the deposit, the type of equipment used and the number of people employed. A generalized version of mine size categories established in the EPA's 1985 placer mining effluent guidelines is used in this manual. Small-scale operations are classified as those processing less than 20 cubic yards per day. Large-scale operations range from 20 cubic yards to 4000 cubic yards per day. Other than large floating dredges, few operations process more than 4000 yards of gravel per day.

Excavating Equipment

Equipment selection and utilization play a large role in the mining method employed. Knowledge of each type of equipment's specific attributes and suitability to perform a particular task is fundamental to understanding mining methods encountered in the field. Excavating equipment at any particular mine may consist of one type or a combination of several types including: bulldozers, frontend loaders, drag lines, scrapers, backhoes, dump trucks, hydraulic giants and dredges. Dredges will be discussed as a separate mining method in a later section. Following is a discussion of the mechanized excavating equipment capabilities and uses:

Bulldozer - It can be utilized in all phases of the mining operation such as overburden removal, excavation and hauling of paydirt, feeding the recovery system, bedrock excavation, road construction, tailings removal and stockpiling, recontouring reclamation and positioning the recovery system. It can excavate, haul and feed the paydirt all in one cycle eliminating rehandling. It can also excavate frozen ground because of its

combination of good breakout force, ripping capability, and traction. Disadvantages are its inability to economically excavate to excessive depths or push material in excess of 500 feet.

Frontend Loader - Its primary use is in hauling previously excavated and stockpiled gravels and loading the recovery plant and removal of tailings. A primary disadvantage is it is restricted to excavation of relatively loose bank gravels because of its limited breakout force. Loaders are either crawler or tire mounted. Tire-mounted units can economically haul material for distances of up to 1000 feet. Their versatility also makes them useful for paydirt and overburden hauling and overburden reclamation work.

Backhoes - The backhoe is a very efficient machine for selective bedrock cleanup. Because it has no capacity for hauling material beyond its swing diameter, it is often used to feed stockpiled gravel or excavating paydirt in conjunction with a portable recovery plant. They are also effective in cleaning slimes from settling ponds, digging drains, digging sumps for pump installations, and working in confined mining areas.

Draglines - Although draglines are not as common as the machinery described above, they are well suited for excavating large quantities of overburden and paydirt, stockpiling material, loading the recovery system, removing tailings and cleaning out settling ponds. With booms up to 70 feet long, they are capable of acting as the single piece of mining equipment. They are often employed with deep deposits containing large reserves and are commonly used in dredging operations. Disadvantages to open cut mining include inefficient bedrock cleanup and inability to handle difficult digging situations.

Scrapers - Scrapers are noted for their speed and high productivity and are primarily utilized for hauling overburden, paydirt and tailings. Because of their limitations in excavating consolidated or unsorted material, a bulldozer is used for initial excavation. The nature of their dumping mechanism also limits their ability to directly feed the recovery system and another type of equipment is usually required for this operation.

Dump Trucks - Trucks may be used for hauling overburden, paydirt, and tailings. Trucks require other loading equipment and are used in large mining operations requiring large volume and long distance hauling.

Hydraulic Giants - Utilizing giants to excavate material was commmon years ago but is relatively rare if not nonexistent today. Their use will not be repeated here except to say that in some operations they are used to remove overburden and sometimes to prewash and slurry sluice feed.

Processing Equipment

After excavation, paydirt is sent through a processing and recovery system to concentrate and remove the gold. The basic principles used to separate and concentrate gold in the recovery system are the same as that used by the stream in the original concentration in the placer deposit namely, natural phenomenon such as gravity, size classification, turbulent fluid flow and specific gravity. Gravels are processed by mixing with water to

create a slurry. Larger gravel is often prescreened and the slurry is sent over the sluice or other recovery device to capture the gold. The finer and lighter material in the slurry continues on and is discharged from the system as tailings.

The most widely used recovery devices are: water pumps, sluices, feed hoppers, conveyor belts, stationary screens (grizzlys), trommels and vibrating screens.

Water Pumps - Water is required to wash the gravel and initiate slurrying of the feed. In the past, water was introduced by ditches, using natural slopes and water pressure. This has largely been replaced by diesel engine centrifugal pumps and plastic, aluminum or steel pipe. Water is pumped to a giant nozzle at the head of the recovery system or through a manifold with several small-diameter outlet nozzles. Water emerging from a manifold system has high discharge velocities and is very effective in breaking down and slurrying the feed material. Pumps are also utilized to recycle settling pond water.

Sluice - Nearly all mining operations in Alaska use the sluice box as the primary concentrating device. Sluice design is usually diverse and may be quite complex. In general, capacities and performance vary with box slope, width, length, gold particle size, physical nature of the feed and availability and amount of water used. Sluices are capable of processing poorly sorted gravels and may also be used for rough concentration in conjunction with other specialized recovery devices. Gold recovery is very dependent on the degree of previous size classification.

Sluices are often used as the only recovery unit and may be situated on or above bedrock. In some operations paydirt is still loaded directly into the head of the sluice without sizing. In most cases, paydirt is loaded through a feed hopper or size classifier.

Feed Hoppers - Hoppers are essentially a tapered box which may feed the recovery system directly or load a conveyor belt which in turn feeds the recovery plant. Hoppers are generally designed to hold enough material to provide a steady, nonsurging flow of gravel to counter the surges inherent in equipment loading cycles.

Conveyor Belts - Conveyors provide a simple and efficient method of transporting large quantities of material over fixed distances. Conveyors also provide a constant rate and volume of feed to the recovery system. They may be portable to accommodate moving around the mine site to follow the progressing excavation of paydirt. Conveyors are also employed to remove and stack oversize material. In this instance, the waste material may be directed into a previously mined cut, eliminating the need for removal by another piece of equipment.

Stationary Screens - Usually known as grizzlys, stationary screens consist of steel bars or rails constructed a fixed distance apart, They may be fixed with a divergent downslope configuration to more efficiently reject oversize material. The screen is set at an angle so rocks will slide off and away from the recovery system. Grizzlys are set directly over the head of the recovery system or feed hopper.

Trommels - Trommels consist of a rotating and slightly inclined cylinder, commonly divided into two sections. The first consists of a scrubber section containing lengths of steel rails fastened longitudinally to the inside of the rotating drum. The rails lift the feed and assist in disintegration and slurrying. The second contains one or more cylindrical screens allowing the fine material to drop through and continue on to the recovery system. The rotational movement assists working undersize particles through the screen.

Vibrating screens - Vibrating screens are used for primary or secondary size classification. They consist of a screen deck or a series of decks for different sizes. The whole unit is vibrated to assist disintegration of the feed and working the smaller material through the screen. Screens consist of woven wire, parallel bars or punched sheet metal.

Small-Scale Operations

More leisure time and interest in outdoor activities over the past two decades, as well as a dramatic increase in the price of gold, has lead increased numbers of people to experiment with placer gold mining and occasionally to go into small-scale production. Gold nuggets have lured the novice to expend considerable effort and money in this quest often with no consideration for earning a profit, but rather for pure personal pleasure and satisfaction.

However, placer mining can be conducted effectively and profitably on any scale. Small-scale operations are usually restricted to shallow gravel deposits in gulches and creeks because of the limited capacity of the excavating equipment and techniques used. The novice or "weekend prospector" commonly uses simple techniques of manual mining such as hand panning, rocker boxes and small sluices.

Small, floating suction dredges are often used. These consist of a pump, suction hose, and a riffled sluice box situated on inner tubes or some other form of floating platform. The operator directs the action of a suction hose to vacuum fine-grained sand and gravel around boulders and crevices or other natural gold traps below the water surface. The material is sent over the sluice to capture the gold. This form of mining takes place in active stream channels and requires clear water so the operator can most advantageously direct the action of the hose.

Small, self-contained washing plants such as the Denver Gold Saver are often used. They can be trailered or hauled by truck close to the mine site. Gravels are usually hand shoveled as typical plants can only handle up to 3 cubic yards of gravel per hour but a small backhoe or frontend loader may also be employed.

Small skid-or tire-mounted, scaled-down versions of larger plants (described in the following section on large-scale methods) are used in more serious production and profit-oriented operations (Figure 11). Smaller equipment may also be used in conjunction with bigger operations to mine small sections of shallow gravel at the margins of the deposit.

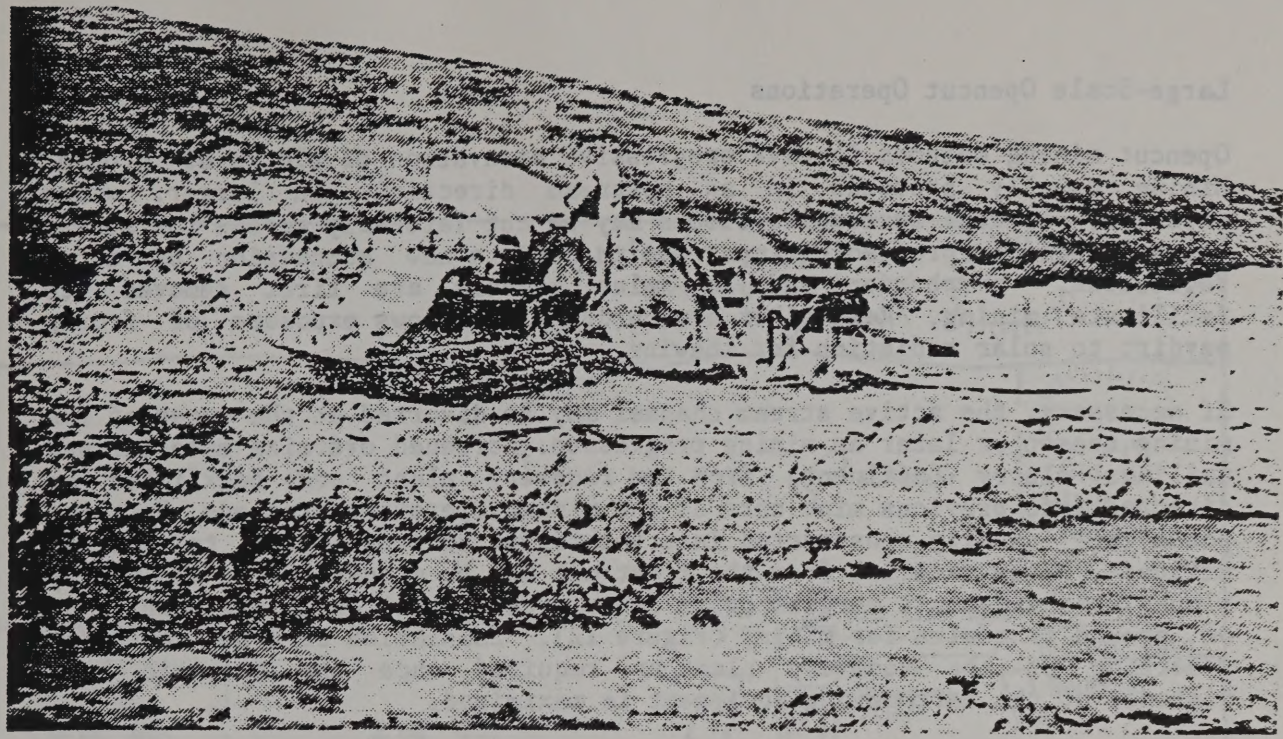

Figure 11.- Small, portable plant mining margins of large-scale operation's mine cut.

Large-Scale Opencut Operations

Opencut mining methods involve progressive excavation of blocks of ground. Mining usually proceeds in an upstream direction. In larger, wider deposits, several successive blocks may be cut laterally across the stream channel before proceeding upstream to begin a new succession of cuts. Vegetation, overburden and low-value gravels are first removed to facilitate mining. Removal of overburden also allows exposure of frozen paydirt to solar radiation for thawing.

If necessary, the active stream channel may be diverted at the onset of the mining season or later as mining progresses. Screened oversize material and tailings slurry (effluent) from the recovery plant are backfilled or directed into previous mine cuts as the mining operation proceeds. Settling ponds are usually constructed in a backfilled tailings area to receive effluent. Waste material from the first cut may be stockpiled on the valley slopes at the deposit margins or in other areas downstream, preferably not on ground scheduled for mining (Figure 12). Initiation of a new cut and resetting of the recovery plant are required once the economic haul distance of the earthmoving equipment is maximized.

Opencut mining methods are: 1. sluicebox on bedrock; 2. sluicebox above bedrock, and 3. mobile, self-contained processing and recovery plants. Classification is made according to the position of the sluice box relative to bedrock with each successive type showing a relative degree of increasing sophistication and efficiency. Mining methods used in each classification are generally the same regardless of the size of the operation and equipment used.

In the sluicebox on bedrock arrangement the sluice is set at bedrock level and a bulldozer usually excavates and pushes the feed into position at the head of the sluice (Figure 13). Giants or spray bars are used to wash the gravels through the sluice. Constant use of a piece of excavating equipment is required for tailings removal, because without continual removal tailings will pile up and choke the outlet of the sluice. This method does not lend itself to preclassification of material or specialized recovery systems and is consequently less efficient in fine gold recovery.

In the sluicebox above bedrock arrangement the sluice and other washing and recovery equipment are situated on gravel ramps or a metal framework. Sluices on ramps are used in conjunction with a sluiceplate which is essentially a steel box tapered on one end to attach to the head of the sluice. The sluice and sluiceplate are commonly fabricated in a single skid-mounted unit. Paydirt is fed over the side or upper end of the sluiceplate (Figure 14).

The elevated sluice is fed by a hopper and is usually equipped with a grizzly or other size classification device (Figure 15). The hopper may be fed by any type of equipment capable of lifting the paydirt. Elevated sluices are often mounted on skids to facilitate moves. The size of these operations varies from throughput capacities of a few tens of cubic yards to several thousand cubic yards of gravel processed per day.

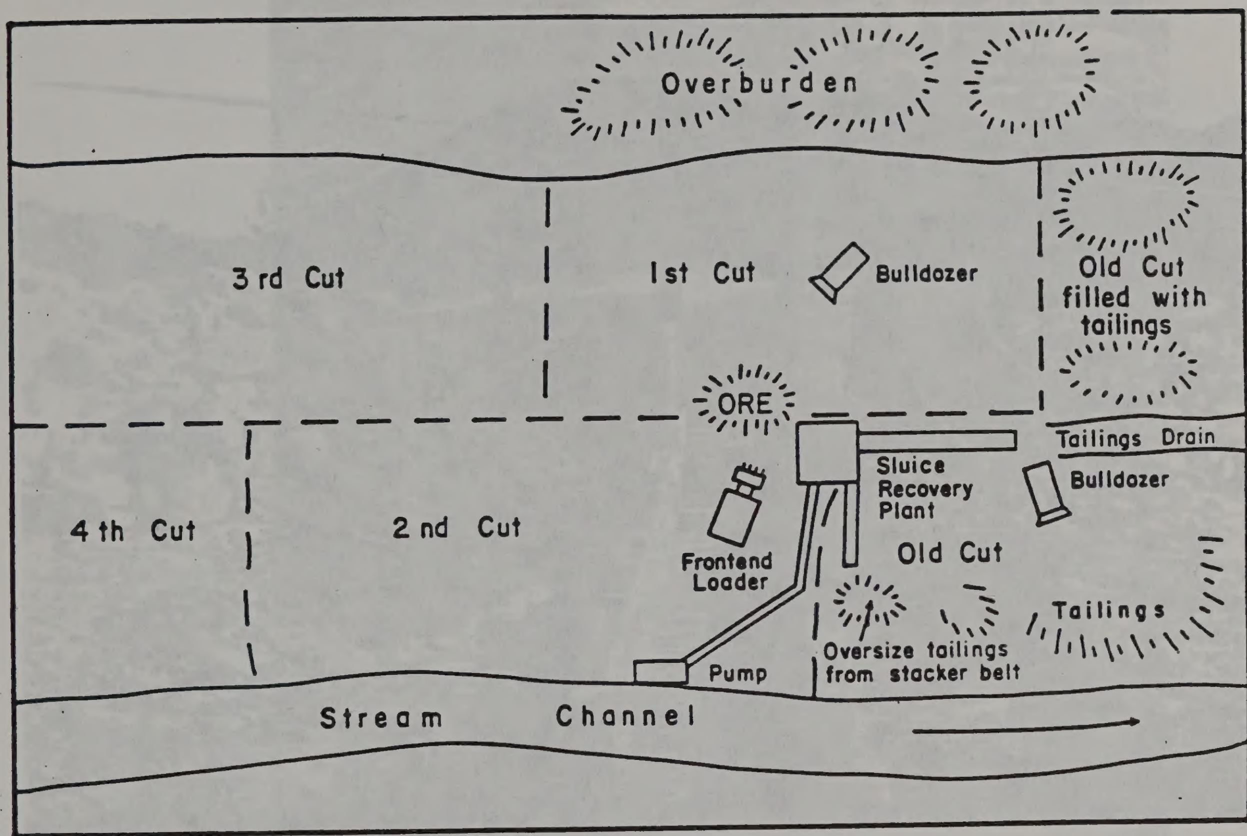

Figure 12.- Possible opencut mine design showing placement of materials and sequence of cuts.

Figure 13.- Wood sluice on bedrock. Note hydraulic spray above wing deflectors at head of sluice. (Photo courtesy of Professor Donald J. Cook.)

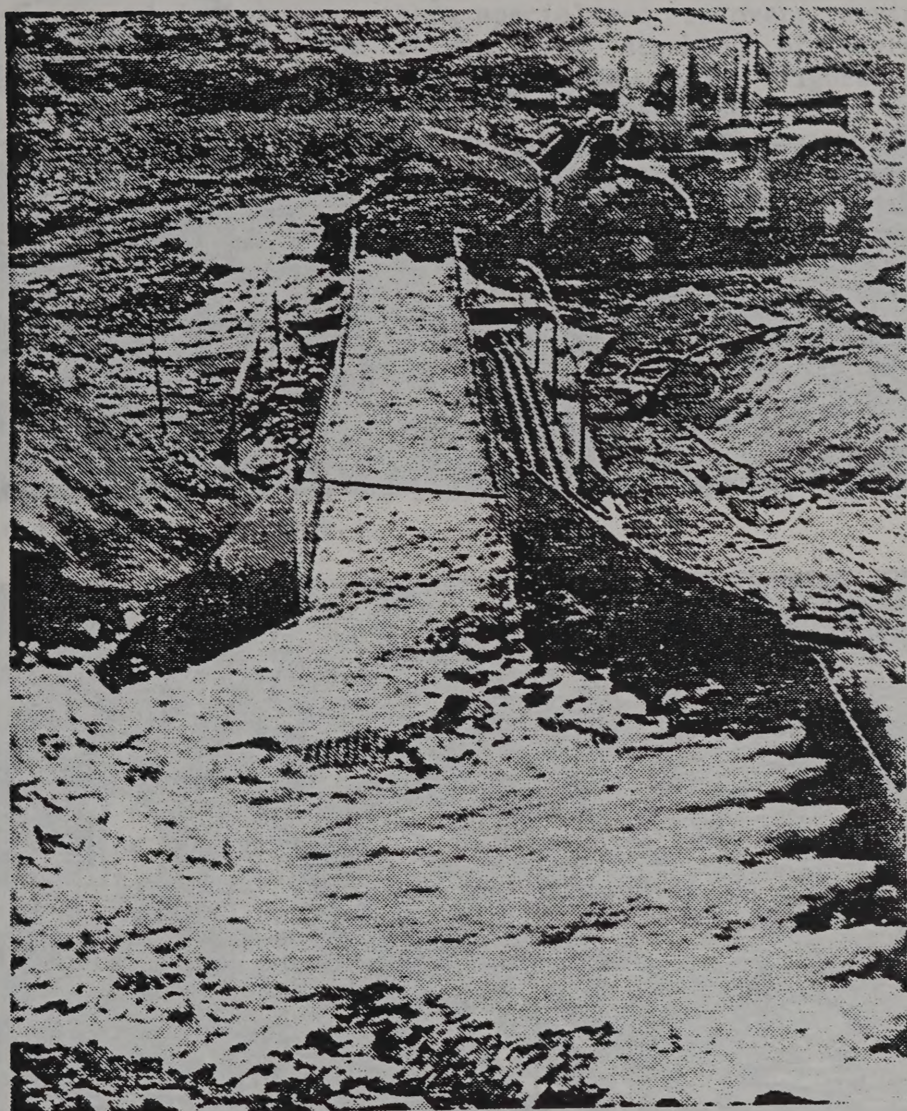

Figure 14.- Sluiceplate feed to sluice. Feed is broken up and slurried with manifold spraybar at right. (Photo courtesy of Professor Donald J. Cook.)

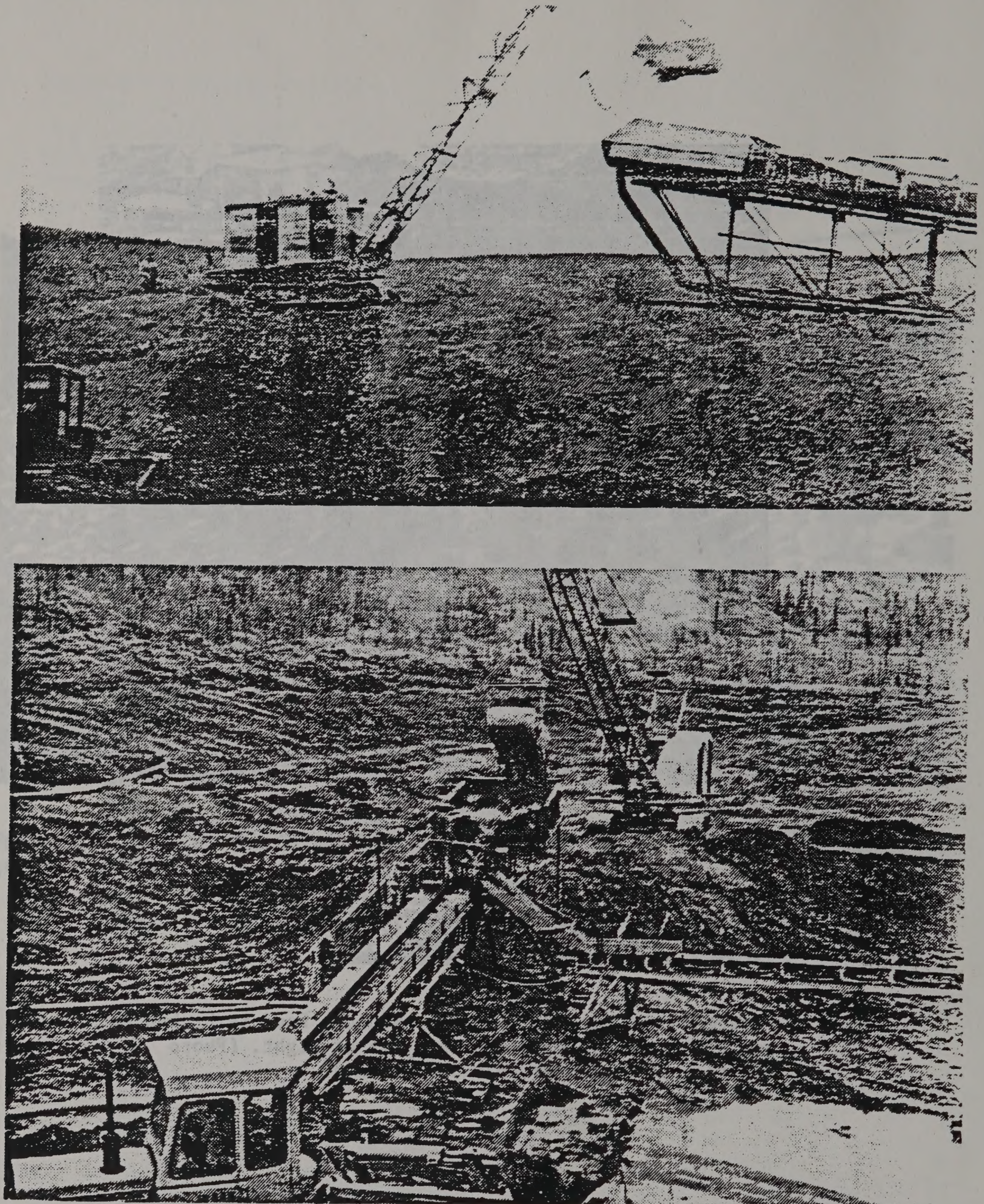

Figure 15.- Elevated sluice fed by dragline. (Photos courtesy of Professor Donald J. Cook.)

The mobile, self-contained processing and recovery plants provide mobility and flexibility increasing efficiency in mining and tailings disposal. The plant consists of a feed hopper, classifying device (usually trommels or vibratory screens), sluice(s) or other specialized recovery devices. Conveyor belts are often used for removal and stacking of oversize material. This system also requires lifting of the feeds (Figure 16). Plants are mounted on tires, crawler tracks, or skids accommodating quick and efficient moves. This mobility facilitates a short-haul distance of the paydirt and also helps to keep tailings handling to a minimum. Plants are often equipped with hydraulic lifts to facilitate quick leveling of the plant after each move.

A recovery plant fed by a backhoe, such as the plant shown in Figure 16, may process up to 100 cubic yards per hour. Long cuts are excavated with the backhoe parallel to the stream direction (Figure 17). The recovery plant is pushed ahead with the backhoe after excavating short (maximum reach of the backhoe) sets in the cut. Oversize material and sluice effluent are directed back into the previous cut, reducing material handling. The first cut is made on one limit of the deposit and doubles as a bedrock drain for mining effluent.

Floating Dredges

Large dredges are employed on relatively flat and water-saturated river, gravel plain, beach, and offshore deposits. Because dredges can process large quantities of material they can economically mine relatively low-grade material. Dredges operating in Alaska use bucketlines to excavate paydirt and feed it to the treatment plant (Figure 18). The mechanism consists of a line of heavy steel buckets on an endless chain supported by a steel-framed dredging ladder. As the bucket-line progressively excavates through a deposit, it carries the pond on which it floats forward (Figure 19). Oversize material is discharged via a conveyor belt stacker. After gold recovery the slurry effluent is discharged back into the pond usually over the stern to backfill. The recovery plant is housed onboard the floating platform.

Dredges are best suited for loose ground with few large boulders. In frozen ground thawing is necessary. Thawing is accomplished by driving pipes (steam points) into the ground and introducing steam or by circulating cool water through a network of pipes and points.

Bucketline dredges have been used in Alaska since 1899. By 1940, 52 dredges were operating. After World War II operating costs made several of the dredge operations unprofitable and only 30 units were reactivated. In 1986 only six dredges were in operation.

Dry Mining and Recovery

Gold has been recovered from auriferous gravels in arid districts, such as the southwestern United States, where water was not sufficient for wet recovery. Other methods were developed which use moving air or wind as the medium of separation. Even under favorable conditions however, a dry concentrator will recover 10 to 15 percent less gold than a wet process (Wilson 1961). This is because the relative weight of gold is about 1.5

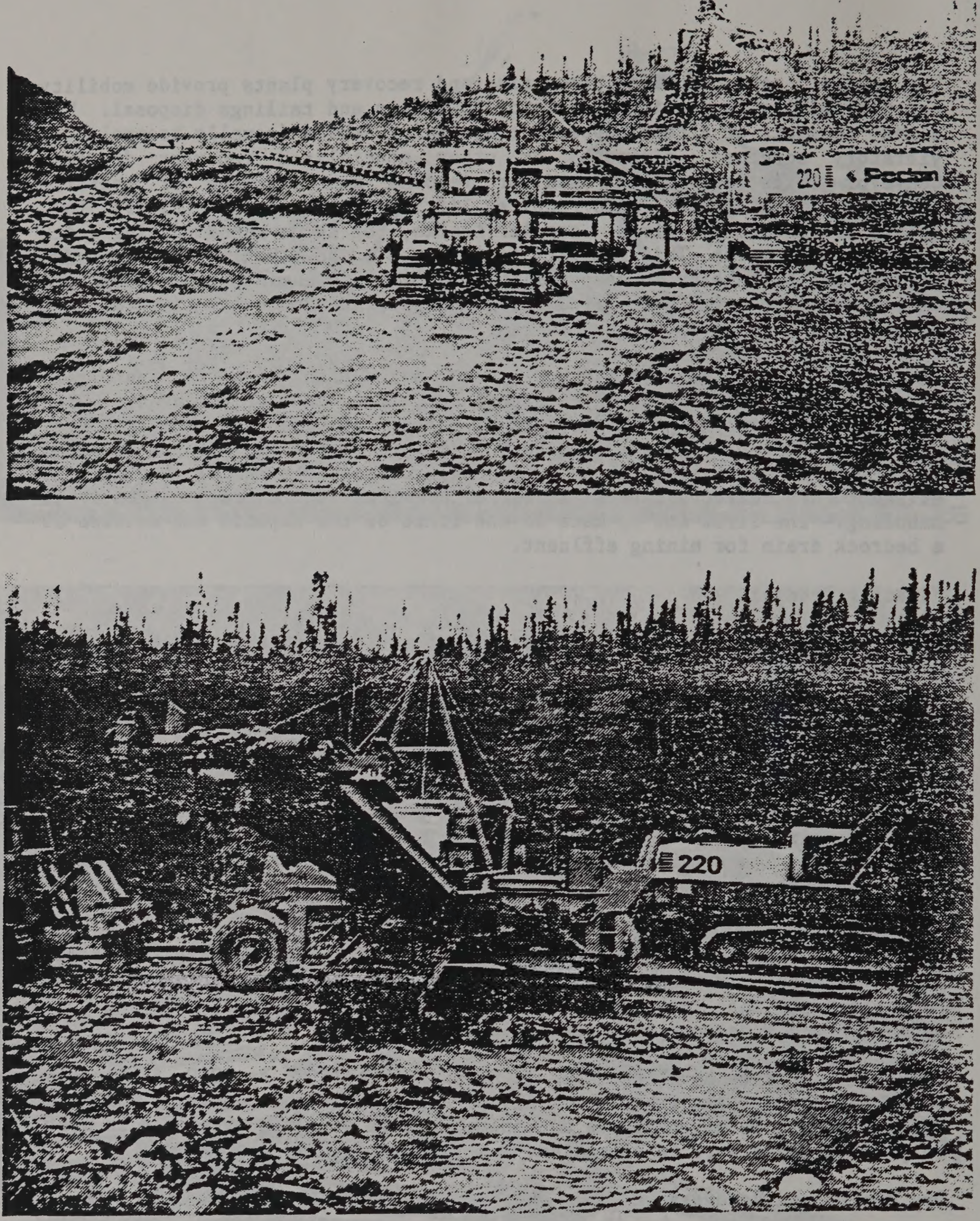

Figure 16.- Mobile, self-contained processing and recovery plant.

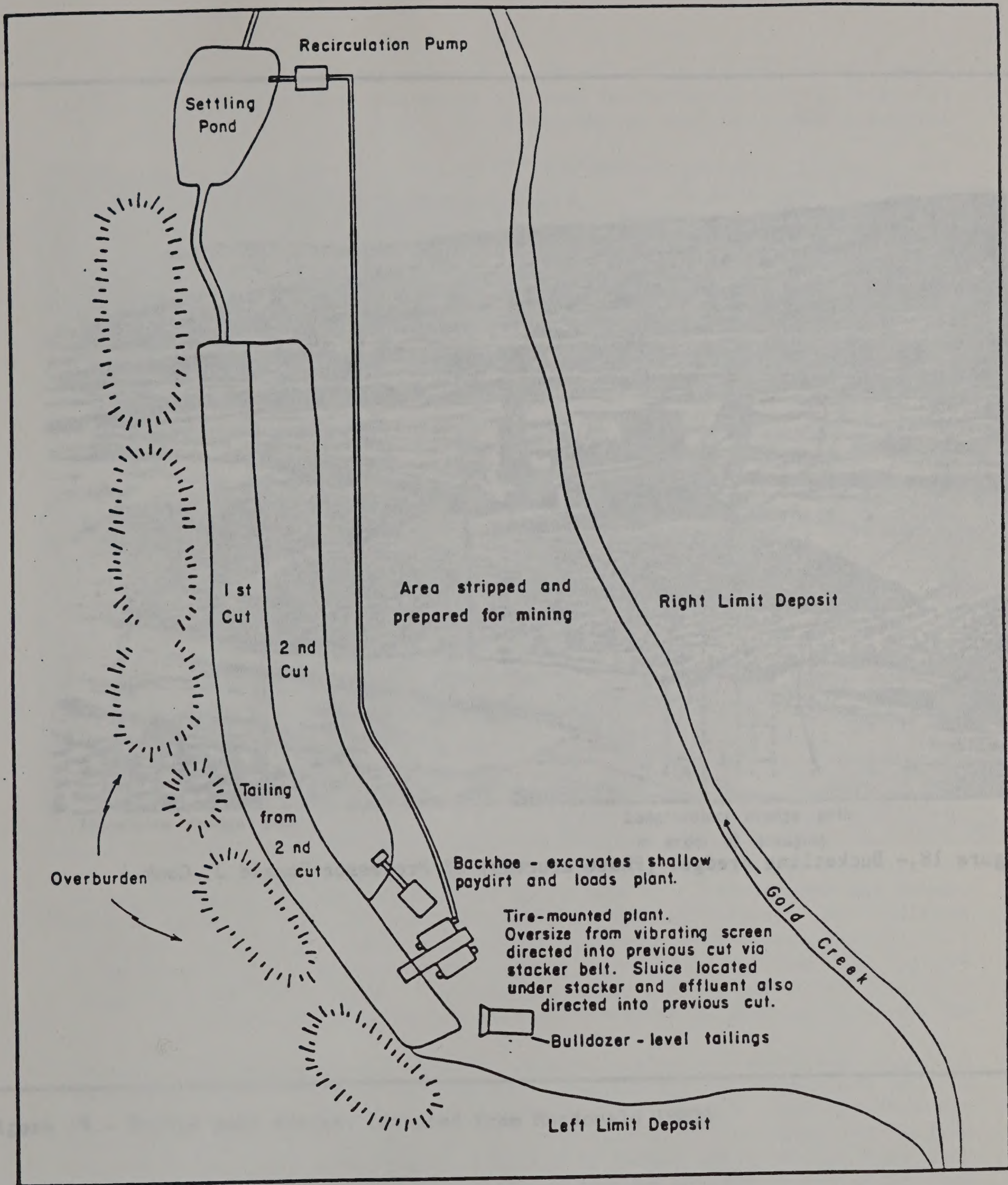

Figure 17.- Possible mining scheme with mobile plant. Sluice effluent travels through first cut.

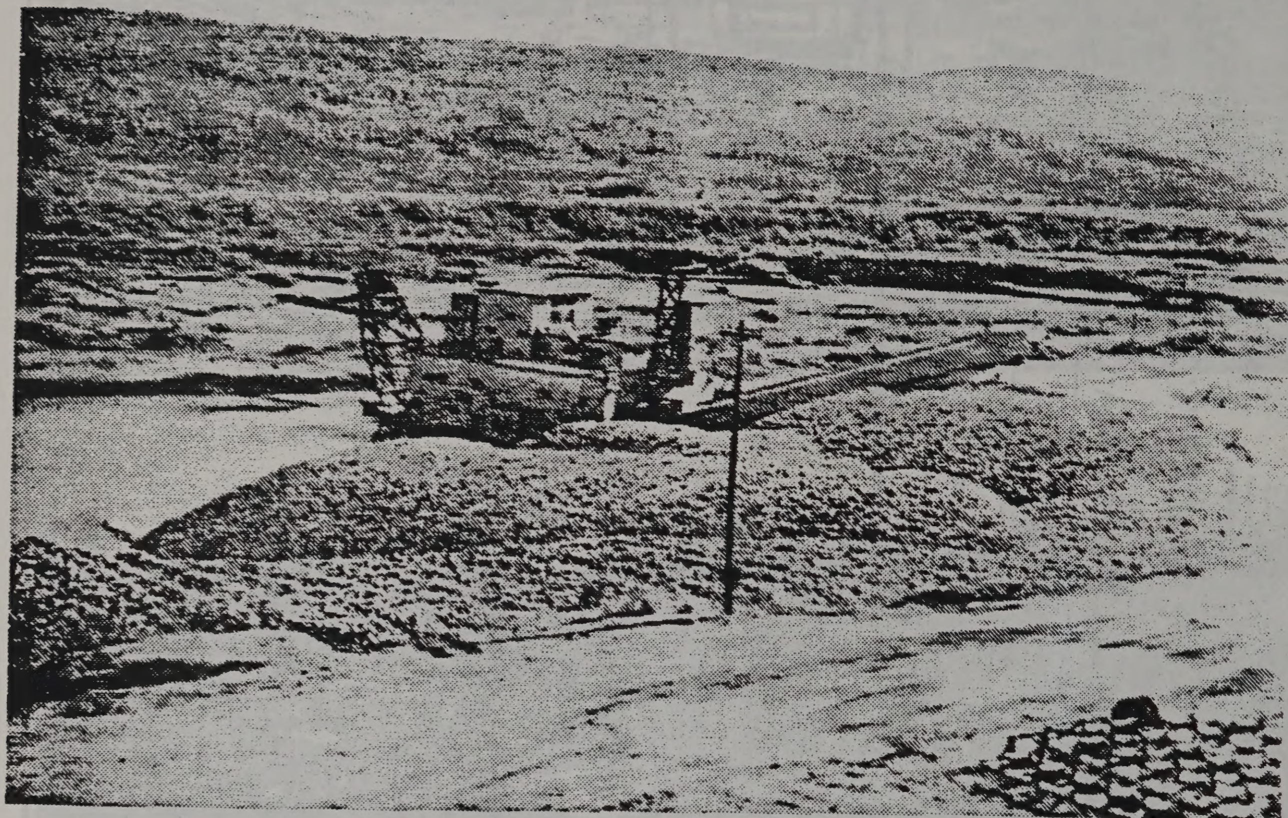

Figure 18.- Bucketline dredge. (Photo courtesy of Professor Donald J. Cook.)

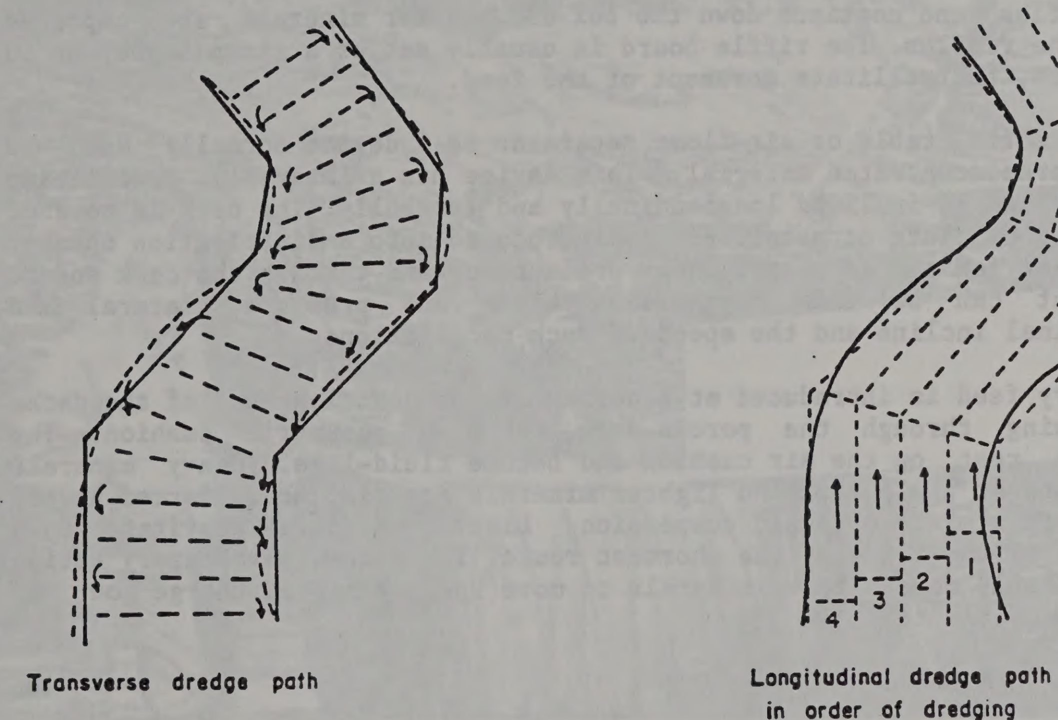

Figure 19.- Dredge path design. (Adapted from Macdonald 1983)

times greater in water than in air. Dry recovery is usually effective on relatively rich deposits and commonly by small-scale methods. Successful large-scale applications are uncommon.

The simplest dry recovery machine is an air jig. The jig consists of a screen for presizing the feed, a feed hopper, a riffle board (sluice) with a wire mesh and fabric floor to allow passage of air and a bellows installed beneath the sluice section. Material flowing on to the riffle board is concentrated by upward pulsations of air through the fabric from the bellows. The intermittent puffs of air cause lighter material to jump the riffles and continue down the box and heavier minerals are captured behind the riffles. The riffle board is usually set at a steep angle, up to 25 degrees, to facilitate movement of the feed.

The pneumatic table or air-float separator is a device normally used to upgrade preconcentrated material. This device has a laterally oscillating deck that can be inclined longitudinally and laterally. The deck is covered with a porous cloth or metal. Air is introduced into a distribution chamber by a blower fan and is passed under pressure upward through the deck cover. Adjustment can be made to the air volume and pressure, lateral and longitudinal incline and the speed of deck oscillations.

As the dry feed is introduced at a corner of the downslope end of the deck, air passing through the porous deck forms a pneumatic cushion. The particles rest on the air cushion and become fluid-like. Heavy minerals concentrate at the bottom and lighter minerals are displaced, forced upward and kept in a state of fluid suspension. Lighter particles gravitate down the slope of the table by the shortest route. The sideways vibratory action of the table causes heavy minerals to move upslope and discharge on the high side of the table.

Gold Cleanup Methods

In sluicing operations, washing and recovery is stopped periodically to remove the concentrate. In operations using more than one sluice it may be possible to divert the flow of slurry from one or more sluices during cleanup and avoid interruption. If jigs or other specialized recovery devices are used in the recovery system, concentrates can be removed on a continual basis and may not require shutdown. Depending on the care taken during cleanup, loss of previously captured fine gold can occur.

Although individual technique varies, sluice cleanup begins with careful washing and removal of the riffles, starting at the head of the sluice(s) and working down. A tank or other receptacle is placed at the tail of the sluice to collect the concentrate. If carpets are employed, they are carefully rolled up and washed in the tank (Figure 20). The concentrate is then subjected to various methods of concentration.

Hand screening or multiple-deck vibrating screens may be used to produce various size fractions to facilitate more efficient concentration of the gold. As the concentrates commonly contain abundunt magnetite, hand magnets or low-intensity magnetic separators may be used to remove the magnetite and reduce the volume.

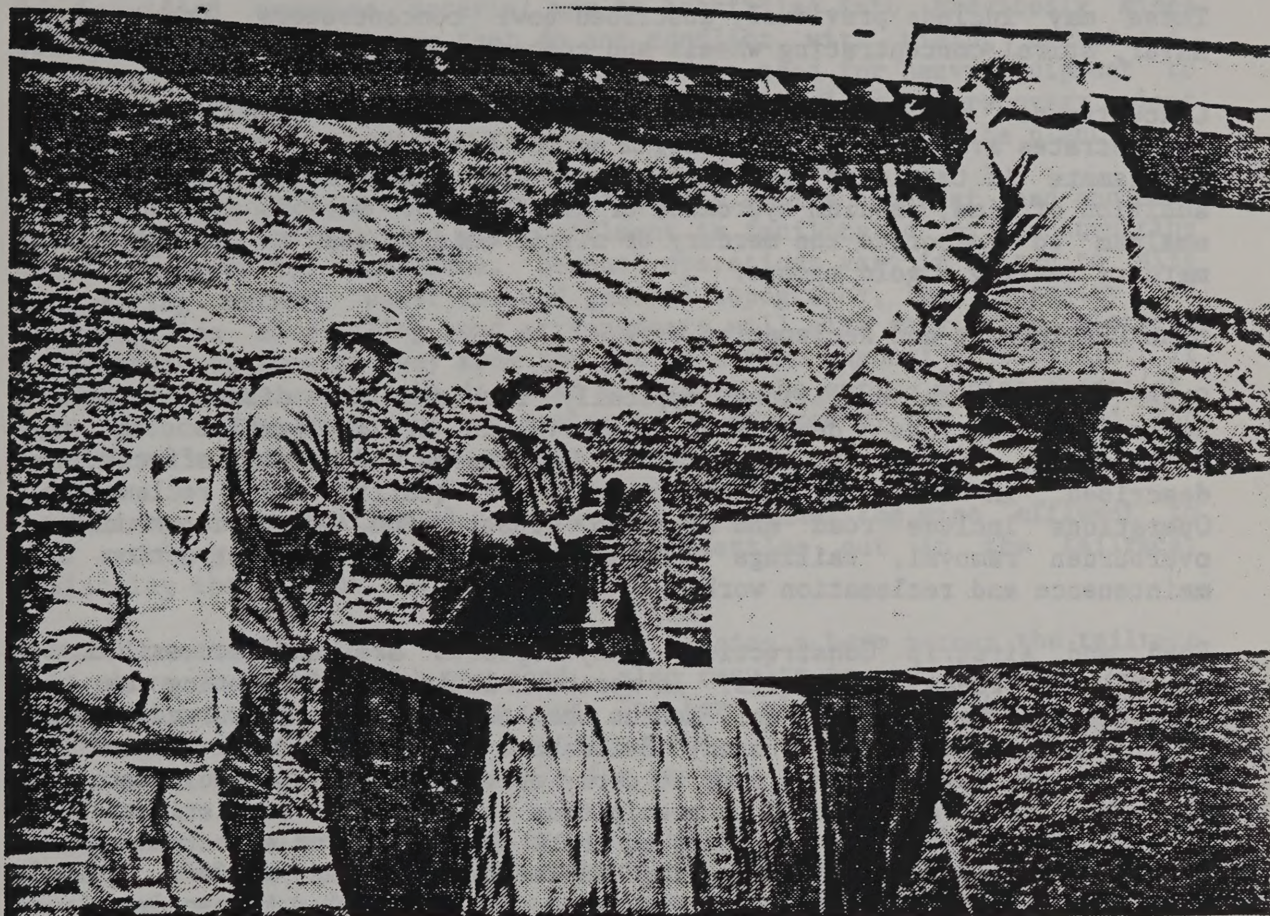

Figure 20.- Washing indoor-outdoor carpet at cleanup time.

Separation of the gold in some operations is still performed by a combination of hand panning or miniature sluices. The concentrate is then dried and hand picked or gangue minerals are winnowed out by blowing on the concentrate. Currently, more sophisticated recovery devices are often used. These may include previously described bowl concentrators, Denver Gold Saver, spiral concentrating wheels and concentrating tables.

Concentrates are commonly amalgamated to recover fine gold. Wet concentrates are rolled in an amalgam barrel with mercury. The gold may not amalgamate if tarnished with magnesium or iron oxides and may require the addition of lime, sodium hydroxide or nitric acid. Heat is applied to the amalgam to evaporate the mercury or nitric acid is used to dissolve the mercury leaving a gold sponge.

Mine Development and Reclamation Work

Other earthmoving operations typically required are discussed below. Several operations deal with preservation or reclamation of the environment. Details of environmental procedures and their effects are described in following chapters on water quality and reclamation. Operations include road and airstrip construction, camp construction, overburden removal, tailings disposal, settling pond construction and maintenence and reclamation work.

Road and Airstrip Construction - Many deposits are not accessible by established roads or trails. In northern latitudes freighting may be conducted in the spring or fall months when tundra can be traversed because it is snow-covered or frozen. Bulldozers or other track-mounted or balloon-tired vehicles are used to carry or tow freight and equipment. If roads have to be constructed in precipitous terrain, the sophistication of construction will vary depending on local conditions and expected amount of use. In addition to excavating and leveling a road surface, gravel surfacing and ditch construction for water runoff may also be necessary.

Some operations use fixed-wing air support requiring a landing strip near the mine. Landing strips are often constructed by leveling old tailings piles or prestripped areas scheduled for mining. In other cases, nonmining areas near the mine are stripped and leveled.

Camp Facilities - Personnel live on site at most mines and space is required which will not conflict with mining activities. Space for equipment maintenance and cleanup facilities also has to be allowed for in minesite planning. Old tailings areas or areas prestripped for mining may be used. In some cases, nonmining areas near the mine will be stripped and leveled for this purpose.

Overburden Removal - Removal and stockpiling of overburden and low-value gravels is required because they are uneconomic to mine and process. Frozen ground and often a thick horizon of fine, muddy silt or "muck" may be present in northern latitudes. Muck is usually frozen and covered by an insulating vegetation cover of moss, brush and trees. The vegetation layer and successive layers of muck are excavated to expose the underlying material to solar radiation for thawing. In other areas gravels may be thawed and require removing only a thin layer of soil and vegetation. If

topography allows, overburden and low-value gravels are placed on valley slopes and away from mining operations. Old tailings areas may also provide space for stockpiling.

Tailings Disposal, Settling Pond Construction and Maintenance - Preclassified oversize material may be backfilled into previously mined cuts or into other areas that do not conflict with mining operations. Conveyor belts are often used eliminating the need for heavy equipment to haul the oversize material. Coarse sand and gravel in the recovery plant effluent may require periodic removal and hauling away from the plant.

Settling of a large portion of the settleable solids (coarser sand and silt particles) in the recovery plant effluent is facilitated by constructing settling ponds and a tailrace. Because operations vary in regard to size and mining method, water use and general topography, the capacity and configuration of settling pond arrangements will also vary. The settling pond should be located away from the active stream channel if possible. If necessary the stream may be diverted to bypass the entire mining operation. This reduces the amount of wastewater to be treated by the ponds and the likelihood of washouts if the stream floods.

A drainage ditch (tailrace) is excavated to channel the mine effluent to the settling pond. Much of the sediment settles out in the tailrace, prolonging the life of the pond.

A small presettling pond, constructed by placing a berm across the tailrace just above where it enters the pond, also results in deposition of coarser sediments prior to entering the pond. The berm also spreads the flow evenly across the entire width of the settling pond. Material settled behind the berm has to be excavated frequently for maximum effectiveness.

Less water used in the mining process results in less wastewater to be treated. Settling pond size can consequently be reduced to achieve a given level of sediment removal. Reduced water volumes may be accomplished by recirculating settling pond water and using classification equipment. A discussion of water quality treatment technologies is presented in chapter 3 of this manual.

Reclamation Work - Reclamation is an integral part of mine planning. It is commonly ongoing as mining progresses. Tailings that have been directed into mined cuts can be graded and recontoured to natural topography. If tailings have been stockpiled, hauling of material to backfill mine cuts may be required. Coarse tailings may be spread over abandoned settling ponds to stabilize the surface and protect the slimes from erosion or the slimes may be removed and used later as a surface cover. Stockpiled low-value gravels, soils and vegetation may be spread over the deposit in their natural order. Organic material may be mixed with the soils to provide a cohesive, stable surface for establishment of new plant growth. Stream channel construction also requires additional excavation and materials handling activities. A discussion of water-related placer mining regulations is presented in chapter 3 of this manual and reclamation is presented in chapter 4.

Denali National Park and Preserve

The Kantishna mining district is located in the park. From discovery in 1903 and up to 1983, the district produced about 78,000 ounces of placer gold. Mining activity increased along with the price of gold in the latter half of the 1970s. Improvements in mining methods and recovery systems have increased placer gold production from 800 ounces in 1974 to an estimated 7500 ounces in 1983. Seventeen placer mining operations were active in 1983. The number of operations decreased by at least 4 in 1984 and by 1985 only 9 mines were active (Griffiths, 1986). At the end of the 1985 season, mining was stopped temporarily by court injunction.

Operations vary from small suction dredges to several large recovery systems capable of processing over 1500 cubic yards per day. Most operations use portable, integrated washing and recovery plants supported by some combination of bulldozers, backhoes, and frontend loaders. Several large, tire-mounted plants (5 in 1983) are largely responsible for the increased production since the mid-1970s because of their large volume capacity, portability, integrated gravel classification devices and other improvements to the sluice which enhanced fine gold recovery.

The district is noted for its coarse jewelry gold recovered from shallow gulch and creek deposits. In 1983, however, 70 percent of the district production was less than 14 mesh in size. Fine gold was largely produced by the mobile plants operating in the lower reaches of the drainage systems and on bench deposits. In one instance, 86 percent of the gold recovered in a large cleanup was less than 20 mesh in size.

Combined capacity for the 17 operations in 1983 was estimated at 11,500 cubic yards per day with a seasonal maximum capacity of 800,000 cubic yards per annum. It was estimated by Levell (1984) that the district contained 43 million cubic yards of minable stream and bench gravels of which 18 million cubic yards were covered by existing mining claims. At the maximum production rate, over 20 years would be required to mine the reserves under claim.

Other than mining activities, roads have been constructed in the recent past to freight equipment and supplies into some of the least accessible drainages. In other cases, existing trails have been used to freight across the tundra. Heavy equipment is largely transported across roadless tundra during late spring when the surface is still largely frozen. Airstrips and camp facilities are usually situated on old tailings. In some cases camps have been set up in areas prestripped for mining or on flat areas above the stream channel. Reclamation work and water quality precautions in 1983 varied from none at all to operations that employed systematic procedures of tailings backfill, recontouring tailings, replacement of overburden, and construction and maintenance of settling ponds. In 1985, at least one of the larger operators successfully employed recirculation of settling pond water.

Wrangell-St. Elias National Park and Preserve

The Nizina and Chisana mining districts are located in the park. The Chisana district includes the summit areas and north slopes of the Wrangell Mountains. From 1913 to 1969, the Chisana district produced about 50,000 ounces of gold from small-scale mining of stream and bench gravels in the Bonanza Creek area. Remoteness of the area, shortages of water on some streams and the small extent of most deposits precluded the development of large-scale operations (Cobb, 1973). During 1985, five small-scale sluicing operations and one small suction dredge were active in the Bonanza Creek area about 20 miles east of the old town of Chisana (Griffiths, 1986).

The Nizina mining district, located largely on the south slopes of the Wrangell Range, was discovered in 1901. Since that time, intermittent but mining of stream and bench deposits has continued. According to Griffiths (1986), during the 1985 season two grizzly-sluice operations were active in the May Creek area. These operations possibly had a throughput of up to 500 cubic yards per day.

Yukon - Charley Rivers National Preserve

The Circle and Eagle districts are located in this part of the Yukon River drainage. Placer gold has been mined in this area since the mid-1890s, with total production from the two districts of about 780,000 ounces through 1960. According to Griffiths (1986), one sluicing operation with a potential to process up to 500 cubic yards per day was active in 1985 in the Woodchopper and Coal Creek area. Mining activities were apparently restricted to prestripping overburden. Some assessment work was also conducted in an area east of the Charley River.

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CHAPTER 2

ENVIRONMENTAL IMPACTS

INTRODUCTION

The adverse impacts created by placer mining are well-documented and often obvious. Public attention has been drawn to areas in California, Colorado and Alaska severely altered by past placering activity. Without laws, regulations and a unified public conscience, past placering proceeded with impunity. Even today the reluctance of many placer operators to invest in less environmentally destructive (and often more efficient) methods of placer mining has resulted in an unfavorable public view of the placer industry. This is a detriment to those operators who strive to operate as efficiently as possible with the least impact to the environment.

Most operations within units of the National Park System in Alaska are on streams and rivers. This results in adverse impacts on both the stream system itself as well as the surrounding terraces and upland areas. The greatest impacts are from the increase in sediment load and turbidity created by activity within the streambed and by erosion of the bank, upper terraces and upland areas due to the stripping of overburden, removal of top soil and destruction of tundra. These impacts can be mitigated by properly designing settling ponds to promote settling of silt and clay-sized particles prior to the release of process water downstream.

In general, the environmental impacts of placer mining involve chemical, physical and biological changes in the stream environment. Not only is the stream in the immediate vicinity of a mining site adversely affected, but also the biological structure, water quality and stream hydraulics may be altered for many miles downstream, depending on the effectiveness of treatment facilities. The size of an operation, as determined by the number of cubic yards of paydirt processed per day, is not necessarily correlated with the amount or degree of adverse environmental impacts. Although large operations expose more land area to potential erosion and process more pay dirt, the amount and degree of downstream impacts are predicated on the efficiency of the mining operation and the effectiveness of treatment.

BACKGROUND

STREAM TYPES

In general, running water systems in Alaska include mountain, glacial, springfed, and lowland streams and large rivers. Placer mining operations in Alaska are typically situated on alluvial gold deposits along headwater tributaries of mountain streams.

Mountain streams are relatively swift and shallow, having steep gradients, straight channels, high velocities and short response times to local precipitation and surface runoff. In areas where the surrounding vegetation is sparse, peak discharge is short and high. Many headwater mountain streams dry up during late summer and freeze to the bottom in winter. A large percentage of the annual discharge occurs over a short period during spring breakup. Flow then decreases throughout summer, except for freshets during periods of occasionally

heavy precipitation. By early fall, the smaller of these streams freeze to their beds. Larger mountain streams tend to have wider, deeper, and more sinuous channels, with well developed riffle and pool topography. Groundwater recharge in the larger of these streams is often sufficient to maintain some level of flow under the ice throughout the winter.

AQUATIC HABITATS

Stream habitats are very heterogeneous because of variations in physical and chemical composition and aquatic species distribution. Aquatic organisms have characteristic adaptations or preferences with regard to water depth, water velocity, bank cover, chemical quality, temperature and substrate type. These factors limit the distribution, abundance and biomass of stream organisms.

Channel Structure and Hydrology

The rate of sediment transfer is directly related to the rate of energy expenditure by a stream as it flows from a higher to a lower elevation. This is known as unit stream power or USP. Reductions in suspended sediment concentrations occur in meandering streams and streams having natural pool and riffle topography relative to straight channels. These features serve to reduce USP and therefore the erosive energy and sediment transport capability of stream discharge. Stream channelization often associated with placer mining alters the natural configuration of stream channels, thereby increasing the potential for sediment transport, bank erosion and downstream flooding.

When the morphology of a stream channel is modified, stream biota may be affected as a result of increased sediment loads, disruption of aquatic food webs and decreased habitat diversity (depth, bottom type and velocity). As the stream channel becomes straighter (less sinuous), habitat diversity decreases and invertebrate concentrations decrease because of an increase in invertebrate drift. Consequently, there is a direct relation between habitat diversity and the diversity of fish populations.

Ideally, streams should contain riffles and pools in equal proportions. Riffles support high invertebrate productivity but usually provide poor fish cover and are unstable with regard to fluctuations in discharge. Pools, on the other hand, provide better cover for fish, support larger and older fish and are more stable with regard to fluctuations in discharge. Fish may also obtain cover from boulders, overhanging banks and woody debris.

Substrate

Substrate stability is critical to the development and maintenance of healthy stream communities. Very small and very large material are the most stable, whereas intermediate-sized particles are the least stable. In general, stream substrate habitats may be classified as either erosional or depositional. Erosional habitats correspond to portions of the streambed having large substrate particles. Under natural

conditions deposition tends to occur only during periods of low discharge. Thus, small particles are present in the water column only at certain times of the year. In contrast, depositional habitats are composed of relatively small, transient particles which are easily transported by small fluctuations in flow. Stream channel configuration largely determines the distribution of erosional and depositional habitats. Higher velocity headwater streams tend to have larger substrate particle sizes characteristic of erosional habitats. Many other natural streams are a mosaic of these two habitat types.

Benthic organisms exhibit substrate adaptations and preferences according to current. Heterogeneous substrates support higher diversities than homogeneous substrates. As the bottom type becomes simplified by the deposition of sediments from placer mines, the number of organisms decrease. Attached algae are most abundant on cobbles and boulders, less abundant on sand, silt and clay, and least abundant on gravel. Furthermore, the density of benthic organisms generally decreases as the substrate becomes more homogeneous because less habitat space is available. The exception to this occurs when only cobbles are present because they provide the most favorable habitat.

Benthic organisms and other aquatic invertebrates are directly affected by the presence of fine sediment in the water column and the substrate. Significant reductions in aquatic macroinvertebrate populations have been noted in different streams as a result of increased turbidity and suspended solids levels (Herbert and Merkens 1961; Gammon 1970; Sorensen et al. 1977; LaPerriere et al. 1983; Wilber 1983). Elevated suspended solids concentrations also result in increased invertebrate drift (Gammon 1970), changes in species composition (McCart et al. 1980) and decreased feeding success (Arruda et al. 1983; McCabe and O'Brien 1983).

Sediments that settle on the substrate have a blanketing effect on the benthic community and may intrude into stream-gravel interstices. These processes tend to decrease the availability of dissolved oxygen to benthic invertebrates, reduce or eliminate food resources in the form of periphyton and abrade or entrap organisms in the substrate.

Aquatic Plants

Algae may be attached (periphytic) or unattached (planktonic) in streams and rivers. Periphytic algae production predominates in small to mid-size streams, the size used for placer mining. Plankton dominate only in the largest clearwater rivers. Macrophytes represent emergent or submerged vascular plants growing along stream margins, in shallow pools and in riffles. This group of plants provides important substrate habitat for aquatic invertebrates. Macrophytes also provide cover for fish from birds and for invertebrates from fish. However, macrophytes are not particularly important in headwater streams in Alaska.

Theoretically, high latitude streams are primarily dependent on photosynthetic (autochthonus) plant production as the energy base which

drives the system. However, terrestrial (allochthonus) organic material such as leaf litter may be more important than primary production in some streams at certain times of the year. The relative importance of these energy sources in Alaska streams is presently unknown. Primary production is usually the most useful indicator of the availability of energy to the next trophic level, the aquatic macroinvertebrates. Small rivers possess variable growths of periphytic algae depending on levels of turbidity, the scouring effect of current, the abrasive and blanketing effects of suspended sediment and the availability of nutrients. Recent studies in Alaska show that an increase in suspended sediment and turbidity levels from placer operations result in decreased periphytic algae growth and primary production downstream.

Fish

The life cycles of both fish and their prey are greatly extended at northern latitudes so that low production rates can be expected. In general, fish species diversity is positively correlated to the diversity of stream depth, velocity and substrate type. Habitat requirements differ among fish species and change with stage of development. Since young fish (fry) are small and have limited swimming capacities, they hide in the substrate or seek low velocities (boundary zone swimming). Juvenile fish can withstand a broader range of velocity, and adults a broader range than juveniles. Consequently, different sized individuals can occupy different areas of a stream. A lack of habitat diversity often results in mined streams without habitat reclamation.

The major components of fish habitat altered by mining are channel structure, stream discharge and velocity, water quality and the availability of food. Channel structure includes factors such as channel geometry, substrate and streambed materials and objects used as cover by fish. Placer mining typically increases turbidity and suspended solids levels in adjacent streams, thereby reducing light penetration and in turn, fish food production. Feeding ability is also reduced in sight-feeders, such as most salmonids.

Fish can be divided into two general catergories: anadromous and resident. The species comprising these two categories have different habitat requirements. Anadromous fish, such as salmon, migrate to the sea to feed and return to fresh water to spawn. The most prevalent anadromous fish species in placer mined streams in Alaska include the five species of pacific salmon - sockeye, humpback, coho, chum and chinook. Spawning migration usually occurs in spring, summer, fall, and early winter, and is species, drainage and site specific. Successful migration of salmon depends on the hydraulic conditions of the stream such as depth and velocity relative to the swimming capabilities of different sized fish. Placer mining may inhibit passage of migrating salmon by altering the hydraulic conditions of a stream or by imposing physical barriers in the stream channel.

Salmon select spawning habitat with specific depth, velocity, substrate type and intergravel flow characteristics. Spawning salmon use a

variety of water depths ranging from 0.3 to 10 feet. However, spawning salmon are always more sensitive to velocity than depth. The optimal velocity range is between 0.5 and 2.5 feet per second.

Salmon spawning behavior includes the physical construction of a spawning-nest depression (redd), in which the eggs are laid and subsequently covered by rapid body and tail motion. Suitable spawning bed materials range from small gravels to cobbles; larger salmon are able to use larger subtrate material. Placer mining typically contributes high concentrations of fine suspended particles which may be deposited in spawning beds at considerable distances downstream resulting in high embryo mortality because of reductions in oxygen supplies and metabolic wastes removal. High embryo mortality also occurs when the spawning bed is dewatered or frozen.

The Arctic grayling (Thymallus arcticus) is probably the best example of resident fish occupying a variety of habitats in Alaskan streams subject to placer mining. Other resident fishes that may occur in such streams include the whitefishes, such as inconnu, Bering cisco, round whitefish, broad whitefish, Alaska whitefish and humpback whitefish; the slimy sculpin; the longnose sucker; and in some areas Dolly Varden; Arctic char; and rainbow trout. Dolly Varden and Arctic char are anadromous in some areas.

Since grayling are widely distributed and occupy a variety of habitats, they are representative of resident fish affected by placer mining. Upstream migration begins at spring breakup. Because of their relatively small size, grayling have lower maximum velocity thresholds than salmon but are not as restricted by depth. Spawning occurs immediately after breakup in mid-May to June. Once reaching their spawning grounds, they select sites with suitable depth, velocity and substrate characteristics. An optimal water depth may be around 1.0 foot and the optimal velocity is likely 0.5 to 2.0 feet per second. Grayling most often use sandy, gravel bottoms for spawning, perhaps because of its prevalence in Alaska streams.

Velocities high enough to cause bedload sediment movement would likely be sufficient to dislodge and wash embryos downstream since they are covered only by a thin layer in the substrate. The siltation of grayling spawning areas during incubation can cause high embryo mortality because of reductions in oxygen supplies and metabolic wastes removal.

After spawning, adult grayling usually move farther upstream and take up residence in pool areas where their entire diet is composed of aquatic invertebrate forms (larvae, pupae and adults) taken from the benthic drift. Adult grayling establish territories in pools with the largest and strongest fish occupying the most advantageous feeding positions at the head of the pool. Increased sedimentation can result in the filling of choice pool areas, thereby reducing preferred feeding habitat.

The grayling is primarily a surface to mid-depth feeder during summer. As sight-feeding fish, they rely on their ability to see aquatic and

terrestrial insects in the water column. Consequently, the clarity of the water is critical to feeding success and growth. While grayling are known to use turbid rivers for migration, they normally do not remain in these rivers and have been found to avoid turbid headwater streams as spawning and rearing habitat. Several investigations in Alaska indicate a possible decline in grayling populations in certain areas as a result of increased turbidity from placer mining.

Demonstrated effects of suspended solids on grayling include abnormal gill histologies (McLeay et al. 1983), gill tissue damage (Simmons 1984), elevated blood glucose and reduced leucocrit values (McLeay et al. 1983), and impaired feeding success (Simmons 1984).

Summary

Sedimentation from placer mining results in the decline of productivity rates at all trophic levels of the stream community including plant, invertebrate, and vertebrate organisms. Major effects of increased sedimentation on fish populations are avoidance, disruption and possibly elimination of normal reproduction. When suspended particles settle, they blanket spawning grounds or eggs and impair or prevent emergence of recently hatched fry.

Water exchange provides for the transport of oxygen and the removal of waste products in spawning beds. Fine sediments infiltrating stream substrates reduce the rate and magnitude of water exchange through stream gravels used for spawning. Mortality of eggs and alevins (newly hatched fish still attached to the yolk sac)increases dramatically as the percentage of fine sediments increases in the gravels.

Salmonids can tolerate increased concentrations of suspended sediment for short periods of time. Long periods of elevated levels of suspended sediments exert a number of sublethal impacts on juvenile and adult fish including tissue damage by abrasion, fin disease, clogging of gills, stress responses, agressive behavior, starvation and decreased ventilatory efficiency. Suspended sediment and turbidity may also cause fish to avoid previously usable habitat, thereby increasing competition for available food and space in adjacent clearwater streams. The presence of high sediment concentrations (suspended or deposited) resulting from placer mining over several years leads to population decreases or elimination of indigenous fish species.

TERRESTRIAL HABITATS

Terrestrial plant communities in mining areas in Alaska may consist of tundra, herbaceous plant understories, a variety of shrubs, coniferous or deciduous tree canopies or some combination thereof. Riparian plants provide shade and habitat for birds, mammals and invertebrates that live near or frequent adjacent streams. Streamside vegetation also has a large influence on determining stream habitat quality. Heavily forested stream sections tend to be cooler in the summer and warmer in the winter than streams devoid of vegetative cover. Data from studies in forest management show that natural vegetation along streams acts to reduce the transport of terrestrial sediment loads.

The efficiency of reducing sediment loads varies with the type and density of vegetation. When the depth of overland flow is less than the herbaceous plant cover height, sediment loads may be reduced by as much as 50 percent because of the filtering action of terrestrial plants.

Removal of riparian vegetation, a common practice in placer mining, may have a number of direct adverse affects on the stream community. Removal of vegetation upstream may result in signficant reductions in invertebrate and fish production in areas where terrestrial (allochthonus) inputs provide important sources of energy for these organisms. Areas which are stripped of deciduous vegetation commonly have low diversities and numbers of aquatic macroinvertebrates because of reductions in coarse particulate organic material (leaf litter).

Results of stream bank vegetation removal include increased erosion and mass soil movement from the loss of soil-stabilizing root structures, less filtration of fine sediments in surface runoff, decreased interception of precipitation leading to more severe discharge rates, less habitat for emerging aquatic insects and changes in terrestrial insect types as fish food. The loss of tree canopy results in decreased litter inputs, increased solar radiation, increased diurnal temperature variation and increased summer temperatures. The removal of streamside vegetation also eliminates shade and cover for resident and migratory fish.

IMPACTS FROM MINING ACTIVITIES

Impact of a typical sluicing operation on terrestrial wildlife habitat is the decimation of tundra, riparian woodland or boreal forest which is similar to adjacent habitat up or downstream. In many areas where placer gold has been discovered, sluicing operations line the entire length of headwater tributaries. This situation effectively eliminates wildlife habitat for several miles along a stream valley. Furthermore, it may cause migratory animals such as caribou to select alternative migration routes. However, resident mammals such as beaver and moose have been observed in and around settling ponds of mining operations. The presence of mining camps may encourage the visitation of bears to local refuse dumps which may result in the elimination of the offending bears.

Extensive site disturbances in riparian, tundra or boreal forest habitats may preclude the complete natural restoration of such areas. Without proper rehabilitation measures, revegetation and use by wildlife may not occur for many tens of years. In stream valleys where extensive mining has occurred over many years, indigenous flora and fauna populations are locally reduced or eliminated.

Potential aquatic and terrestrial impacts previously discussed may result from the mining activities discussed below. Since most of the impact is related to sediment release to the aquatic environment, the amount of impact is directly correlated with those activities that expose soil to water. Hence, the absolute level of adverse impact from access roads and airstrips is less than the adverse impacts of

separating gold from pay gravel because sluicing introduces significantly more sediment into the water. Similarly, operations sluicing different volumes of pay gravel mainly differ in the potential magnitude of their impacts. However, the absolute amount of impact is not necessarily correlated with the size of an operation. For example, a well-planned large operation using water use reduction techniques, an adequate settling pond system, and a clearwater bypass results in less sediment being discharged to the receiving water than a poorly planned small operation with an inefficient treatment system.

ACCESS

Even properly designed and maintained access roads and airstrips may result in soil erosion or increase stream sedimentation. However, the major impact related to access roads and airstrips is the disturbance of soils near streams during construction. Heavy equipment movement across streams increases erosion of the banks, thereby reducing fish cover and causing short-term sediment increases downstream.

Improperly designed or installed culverts pose impassable barriers to the movement of fry and adult migratory fish. If culverts are undersized or improperly placed, water velocities in the culvert may be too high to allow fish to migrate to their spawning grounds upstream. Furthermore, scour pools may form on the downstream end of such culverts, leaving them "perched" during subsequent periods of low flow. In contrast, oversized culverts may prevent the movement of fish during periods of low flow, if the water is too shallow in the culvert to permit their passage.

OVERBURDEN STRIPPING

Stripping of overburden (organic soil layer) exposes large areas of fine mineral soil to potential erosion. Precipitation falling directly on exposed soil will loosen the particles. Stockpiles of organic overburden are also susceptible to the erosive action of precipitation and surface drainage during the first year, but depending on the location may revegetate in 1 to 2 years. Flow over stripped slopes can erode and transport large amounts of fine sediments and organic material into nearby streams adversely impacting both terrestrial and aquatic biota in a much larger area than that created by the placer operation itself. Hydraulic stripping, usually performed only in ice-rich permafrost areas, occurs at only a few mine sites in Alaska. All of the overburden fines are carried downslope placing a larger burden on treatment facilities such as settling ponds.

EXCAVATION OF PAY GRAVEL

After diversion of the active stream channel, the gold-bearing gravels are excavated from the stream channel, floodplain or bench, and moved to the processing area. This activity loosens the fine soil particles from the gravels which, when combined with surface runoff, creates additional sediment laden water. The excavation often proceeds down to bedrock, leaving little or no soil to absorb precipitation and surface runoff, or to support the root structures of terrestrial vegetation

thereby accelerating the erosion process. As the mining operation proceeds upstream, the excavation is commonly partially filled by either tailings or settling ponds.

SEPARATION AND CONCENTRATION OF GOLD

After excavation, the gravel is thoroughly washed with water and the resulting slurry fed to a sluice box, jig or other gold recovery device. The gold is removed by gravity concentration and the slurry is discharged as tailings. Larger materials (boulders, cobbles and gravel) are removed by heavy machinery and placed in large tailings piles. Finer materials (sand, silt and clay) are carried downstream by the effluent from the sluice. This effluent contains high concentrations of sediment and is the major point-source at placer mines.

CONVENTIONAL TREATMENT

The effluent typically flows down a channel, or tailrace, into a series of settling ponds. Much of the sand drops out of the effluent in the tailrace and medium and large silt sized particles are removed by well-designed and constructed settling ponds. Nevertheless, even under the best of conditions settling pond discharges usually contain high suspended solids concentrations (>100 mg/L) and turbidity levels which may be transported a long distance downstream. The degree of impact that pond discharges have on downstream water quality and stream biota is highly dependent on the concentration and magnitude of effluent sediment discharges relative to the dilution capacity of receiving streams.

Unless all water is diverted around a mine site, water in contact with any area disturbed by mining activities will become sediment laden and turbid, contributing to nonpoint-source pollution. Diversion of stream channels around settling ponds may prevent washouts during flood events, results in a lower volume of water being treated and provides a larger volume of clearwater for the dilution of pond discharge waters. In operations employing recycle systems, channel diversion prevents pond washout and provides greater control of the recycle water balance.

HANDLING AND DISPOSAL OF TAILINGS AND FINES

Boulders, cobbles, gravel and large sand particles are normally placed in tailings piles located in mined out areas. Because of the lack of organic soil, tailings are unsuitable for the immediate establishment of pioneer plant species. Moreover, these relatively unstable tailings may contribute sediments to nearby streams in surface runoff after site closure. Fine material is discharged downstream, often to settling ponds. The fines remaining in the settling ponds are sometimes stabilized by armoring with coarse tailings, but often remain susceptible to erosion by surface runoff if the settling pond dams are breached.

ANCILLARY EQUIPMENT AND FACILITIES

The average crew size of Alaska placer mines is six people, with

domestic sanitation and solid waste disposal consisting of outhouses and small landfills situated well away from the active floodplain and groundwater sources. To date, there have been no reported pollution problems from these activities.

Diesel fuel is commonly stored in steel tanks up to 15,000 gallons in capacity and lubricants are stored in drums. Faulty valves, corrosion of drums and poor housekeeping results in spillage. Spillage may also occur during product transfers. Disposal of heavy equipment waste oil is a potential problem if not disposed of properly.

When remote mining sites are abandoned, the camp and even some equipment are commonly abandoned also. In some cases these sites assume historical significance but often remain as a significant reminder of the adverse impact placer mining has on the environment. Abandonment of present day facilities greatly hinders site restoration by making it necessary to remove and dispose of tools, equipment and hazardous wastes, often at considerable additional expense to the reclamation effort. The requirement of reclamation bonds may mitigate this problem.

SUMMARY

The environmental impacts from a typical placer mining operation cannot be overemphasized. Hydraulic mining, booming and other poor mining practices have devastated entire drainage systems. Since organic matter and other nutrients have been washed downstream the land may remain barren for many years. Barren tailings piles in many streams and rivers in Alaska and in the Lower 48 States remain as stark reminders to placer mining activity in the past. It is imperative that field personnel reviewing a plan of operations for placer mining in a unit of the National Park System or monitoring such a plan in the field be aware of these impacts and of the measures which may be taken by the operator to mitigate or eliminate these impacts.

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CHAPTER 3

WATER QUALITY

1.0 INTRODUCTION

This chapter discusses the effects of placer mining on water quality and describes potential problems accruing from the discharge of materials that significantly change natural water quality. The discussion includes physical and chemical water quality parameters, methods for monitoring parameters having an adverse affect on downstream water quality, current methods for mitigating water quality degradation, and major water-related regulatory requirements that must be met by placer mining operations in units of the National Park System.

WATER QUALITY PARAMETERS

In Alaska, water quality standards are the responsibility of the Alaska Department of Environmental Conservation and appear in Title 18, Environmental Conservation, Alaska Administrative Code, Chapter 70, Water Quality Standards (Alaska Department of Environmental Conservation [ADEC], 1985). Designated freshwater uses in these standards apply to all streams, lakes, and ponds and include the following:

Water Supply

(1) Drinking, culinary and food processing
(2) Agriculture
(3) Aquaculture
(4) Industrial

Water Recreation

(1) Contact recreation
(2) Secondary recreation

Growth and Propagation of Biota.

The water quality standards regulate man-made alterations to the waters of Alaska. The water quality criteria (e.g. pH) when used in combination with the water use designation (drinking, culinary and food processing), constitute the water quality standard for a particular water body. That is, Alaska water quality standards consist of the criteria associated with the designated uses for which a particular body of water is protected. In Alaska, all water bodies except the lower Chena River and Nolan Creek (and most of its tributaries) are classified to protect all uses. Hence, the most stringent criterion of the above water uses applies to all water bodies except the two mentioned above. For example, the most stringent criterion for many parameters applies to drinking water supplies. Criteria established for the protection of multiple water uses define acceptable levels for dissolved gas (oxygen and nitrogen); pH; turbidity; temperature; dissolved inorganic substances; sediment; toxic and other deleterious organic and inorganic substances; color; petroleum hydrocarbons, oils

and grease; radioactivity; total residual chlorine; and residues. These water quality characteristics are discussed below.

True color levels remain the same or decrease from natural levels as a result of placer mining. Levels decrease in tundra areas when overburden is removed and stockpiled away from the active stream channel, thereby decreasing the opportunity for surface water runoff to leach tannins and lignins from the tundra vegetation. Radioactive substances and chlorine are not used in the placer mining process. Residues, such as floating solids, debris, sludge, deposits, foam and scum, do not typically occur as a result of placer mining.

DISSOLVED OXYGEN

Dissolved oxygen concentrations vary with temperature, pressure, and somewhat less with biological activity and generally range from 7.5 to 13.5 mg/L during the summer mining season in unmined streams. According to R&M Consultants, Inc. (R&M 1982), dissolved oxygen concentrations exhibit a slight decrease (a few tenths of a mg/L) during sluicing compared to upstream levels and a slight decrease (a few tenths of a mg/L) as water moves through a settling pond. Turbulence downstream from the sluice and pond causes a slight increase in dissolved oxygen concentrations offsetting the slight decrease noted previously. Dissolved oxygen concentrations downstream from placer mining operations are influenced by the organic content of the overburden and the gravels being mined (ADEC 1979). Placer mining operations generally have little effect on dissolved oxygen concentrations (Cook 1979). This is applicable to sluicing inorganic material, but does not necessarily apply to the hydraulic removal of organic muck. Dames & Moore (D&M 1976) report that the dissolved oxygen concentration decreased from 13.4 mg/L to less than 1 mg/L at one mine where organic muck was being stripped hydraulically. It was reported that this rapid dissolved oxygen depletion was caused by chemical oxidation, but similar dissolved oxygen depletion rates were not observed at other operations removing muck.

Saturation of dissolved oxygen is typically 90 percent or higher at placer operations. These saturation levels compare favorably with values in unmined streams. D&M (1986) reports the average saturation level for unmined streams is 96 percent and for actively mined streams it is 99 percent. The slight increase is caused by turbulence as the water moves through a sluice and as it cascades from a settling pond. An exception to this situation could be the hydraulic removal of organic muck. Relatively high concentrations of dissolved oxygen and high percentage saturation values at placer operations moving inorganic material indicate there is essentially no oxygen demand resulting from placer mining.

The Alaska water quality criteria for dissolved oxygen in the water column is most stringent for the protection of freshwater aquatic life (ADEC, 1985). The criteria are: 1. Dissolved oxygen shall be greater than 7 mg/L in waters used by anadromous and resident fish; 2. in no case shall dissolved oxygen above 17 mg/L be permitted; and 3. the concentration of total dissolved gas shall not exceed 110 percent of

saturation at any point of sample collection. Surface waters throughout and downstream from placer mines typically meet these criteria. However, hydraulic stripping of organic overburden may remove dissolved oxygen from the water causing concentrations to fall below the criteria.

Sediment-laden discharges from some placer mines may pack streambeds with fine particles creating low dissolved oxygen concentrations in the interstitial water. Bjerklie and LaPerriere (1985) report mean dissolved oxygen concentrations ranging from 0.3 to 3.0 mg/L in subsurface waters of two streams receiving mining effluent. Neither stream meets the Alaska criterion, which is: "In no case shall D.O. [dissolved oxygen] be less than 5 mg/L to a depth of 20 cm in the interstitial waters of gravel utilized by anadromous or resident fish for spawning" (ADEC 1985).

pH AND ALKALINITY

pH is the hydrogen-ion acitivity of a solution, which is an indication of the solutions acidity or alkalinity. A pH of 7.0 is neutral. pH values commonly range from about 6.7 to 8.0 in unmined Alaska streams. Headwater streams (where most mining occurs) usually exhibit pH values between 6.8 and 7.4. According to R&M (1982), pH decreases slightly during sluicing and in settling ponds. Three reports (D&M 1976; Bainbridge 1979; R&M 1982) were compiled to assess pH levels at various locations throughout placer mining operations. Data from these reports indicate that there are no significant pH changes caused by placer mining. pH levels were shown to display a slight decrease - a few tenths of a pH unit - in the mining process, but returned to normal levels in the receiving water.

Alkalinity is the capacity of a solution to neutralize acids. It is related to the bicarbonate, carbonate and hydroxide content of the water. In Alaska, bicarbonate alkalinity is the dominant form because pH values are normally less than 8.3. Most Alaska streams contain relatively low alkalinity concentrations, ranging from near zero up to approximately 100 mg/L as calcium carbonate, indicating the buffering capacity is low to moderate. According to D&M (1986), the average alkalinity concentration for unmined streams is 16.4 mg/L as calcium carbonate and for actively mined streams it is 31.8 mg/L as calcium carbonate. With a natural low buffering capacity, one would expect to see significant changes in pH levels if placer mining created an impact. The current data indicate placer mining causes no significant impacts on pH or alkalinity levels.

For the protection of freshwater aquatic life the Alaska water quality criteria for pH are: "Shall not be less than 6.5 or greater than 9.0. Shall not vary more than 0.5 pH unit from natural condition" (ADEC, 1985). Based on available information, pH levels are not significantly affected by existing placer mining practices. The Alaska pH criteria can be, and usually are, met at placer mines. It should be noted that pH less than 7 and occasionally less than 6.5 is common within placer mining operations and in many Alaska streams unaffected by human activities. When natural pH is less than 6.5, man-made disturbances

are not allowed to cause the natural pH to vary more than 0.5 pH unit.

The Alaska alkalinity criterion is 20 mg/L or more as calcium carbonate for the protection of freshwater aquatic life except where natural concentrations are less according to the Environmental Protection Agency (EPA 1976). Placer mining operations meet this criterion.

TEMPERATURE

Natural water temperatures range from 0 degrees C up to 15 or 20 degrees C during summer hot spells. Temperatures typically range between 5 and 15 degrees C for most of the summer placer mining activity peak. Water used for hydraulic stripping (which is not common but still occurs in Alaska) can change in temperature depending on whether frozen or thawed overburden is being stripped (ADEC 1979). According to R&M (1982), temperature increases slightly during sluicing and continues to increase slightly in settling ponds. However, Bainbridge (1979) notes that water temperature was not significantly altered when used for sluicing. Furthermore, settling ponds have little apparent effect on temperature according to R&M (1982).

Alaska receiving water temperature criteria (ADEC, 1985) are: "Shall not exceed 20 degrees C at any time. The following maximum temperature shall not be exceeded, where applicable:

Migration routes	15 degrees C
Spawning areas	13 degrees C
Rearing areas	15 degrees C
Egg & fry incubation	13 degrees C

For all other waters the weekly average temperature shall not exceed site specific requirements needed to preserve normal species diversity or to prevent appearance of nuisance organisms." Although there is some small change in water temperatures caused by mining, changes are typically less than natural variations exhibited in Alaska streams during the mining season. Most existing operations do not exceed the Alaska criteria for temperature.

CONDUCTIVITY

Conductivity indicates the capacity of a solution to carry an electrical current, which is related to total concentration of dissolved solids. Conductivity and total dissolved solids levels are typically low in Alaska headwater streams where most placer mining occurs. According to D&M (1986), average conductivity levels in the Crooked Creek drainage are 108 micromhos/cm at 25 degrees C in unmined streams compared to 165 micromhos/cm at 25 degrees C in actively mined streams.

The most stringent Alaska water use category for dissolved inorganic substances is drinking water supply. This category limits total dissolved solids to 500 mg/L, chlorides to 200 mg/L and sulfate to 200 mg/L (ADEC 1985). Placer mining does not cause receiving waters to exceed any of these levels.

NUTRIENTS

Nutrients are substances needed for the growth of plants and animals. Nitrate and phosphate are considered the most critical. Concentrations of nutrients in unmined streams are typically low. Data reported by D&M (1986) for the Crooked Creek drainage indicate that total phosphorus ranges from less than the detection limit up to approximately 0.3 mg/L and nitrate plus nitrite concentrations range from less than the detection limit up to approximately 1.6 mg/L. The ranges of total phosphorus (<0.05 to 0.70) and nitrate plus nitrite (<0.02 to 1.90 mg/L) in mined streams are comparable to those in unmined streams. However, average nutrient concentrations in unmined streams are 0.06 mg/L for total phosphorus and 0.22 mg/L for nitrate plus nitrite, whereas the average concentrations in actively mined streams are 0.15 mg/L for total phosphorus and 0.74 mg/L for nitrate plus nitrite. Two reasons account for the relatively small change in nutrient concentrations at placer mining operations. First, placer mining involves moving primarily inorganic material. Second, miners do not discharge domestic or other organic waste products to their receiving streams.

There is no Alaska receiving water criterion for phosphorus. The Alaska criterion for nitrate of 10 mg/L, is established for the protection of drinking water supply (ADEC 1985). Nitrogen concentrations are within the acceptable limit of the nitrate criterion downstream from placer mines. Furthermore, there is no indication that nutrient levels are increased by placer mining to a level that would adversely affect the aquatic system by promoting eutrophic conditions. Total phosphorus and nitrate plus nitrite concentration ranges in mined streams are comparable to concentrations upstream from mining operations and in unmined streams.

SEDIMENT

Total Suspended Solids

Total suspended solids (TSS) are the portion of organic or inorganic material in water that is retained on a glass fiber filter. According to Peterson et al. (1985), solids in suspension can cause invertebrate drift, cause fish to avoid previously usable habitat, prevent fish from seeing their prey and cause physical damage such as gill irritation to fish. The lethal tolerance of salmonids and other aquatic organisms to suspended solids appears to be relatively high. In most instances, sublethal effects occur at much lower concentrations.

Undisturbed (non-glacial) streams above mining operations typically exhibit low solids concentrations. TSS concentrations are less than 50 mg/L except during spring breakup and summer floods, but are still less than about 100 mg/L during these events. TSS concentrations increase significantly over natural conditions when water is used to wash paydirt through a sluice. Concentrations remain high in the tailrace and settling pond even though a large volume of solids settle in these areas. The highest concentration reported in the literature, 535,000

mg/L, occurred downstream from hydraulic stripping of organic muck (D&M 1976). This is about 10 times higher than the typical TSS concentration found in a sluice effluent. Although operations having settling ponds release higher quality water than operations without ponds, TSS concentrations downstream from ponds are usually about 100 times higher than natural conditions (R&M 1982). Settling ponds do not effectively remove the fine particles, and filtration of mine effluent through tailings dams often allows relatively high concentrations of TSS to pass through.

Data from three reports (D&M 1976; Bainbridge 1979; R&M 1982) were compiled to assess TSS concentrations at various locations throughout placer mining operations. Data were compiled for operations having ponds, but not all of these ponds were properly designed or maintained. TSS concentrations upstream from the operations averaged approximately 20 mg/L, 25,000 mg/L at the sluice effluent, 16,000 mg/L at the settling pond influent, 1900 mg/L at the pond effluent and 800 mg/L at a point 500 feet downstream from the pond effluent or at the confluence with the receiving stream. EPA (1985) reports the monthly average TSS concentration of the pond effluent as 1700 mg/L. This report contains data collected at a number of mine sites over a two-year study period.

The Alaska water quality criterion for sediment, including TSS concentrations, is: "No measureable increase in concentrations of sediment above natural levels" (ADEC 1985). Placer mining discharges typically exceed this criterion.

Settleable Solids

Settleable solids are solids in suspension that will settle under quiescent conditions in one hour in an Imhoff cone due to the influence of gravity. According to Peterson et al. (1985), settleable solids have direct and detrimental effects on aquatic biota and habitat by smothering fish eggs, alevins and invertebrates, reducing intergravel flow, and by coating aquatic vegetation, thus reducing the potential for photosynthesis.

Settleable solids levels are consistently less than 0.1 mL/L (the detection limit of the method) in natural clearwater areas upstream from placer mining. They are significantly higher in a sluice box effluent - on the order of 70 mL/L, remain high in the tailrace - about 50 mL/L, and are low to moderate ranging anywhere from <0.1 mL/L up to 5 or 10 mL/L at the settling pond effluent depending on the effectiveness of the pond. As a result of a two-year study at a number of mine sites, EPA (1985) reported monthly average pond effluent settleable solids levels of 0.2 to 0.5 mL/L. Although operations having ponds release higher quality water than operations without ponds, R&M (1982) found that settleable solids levels at operations with ponds were usually about 10 times higher than natural conditions, or about 1 mL/L. However, EPA (1977) states that it appears that settleable solids can be reduced to or below 0.2 mL/L by allowing four hours of quiescent settling in a pond having a minimum depth of five feet. Well designed and maintained settling ponds are capable of consistently reducing settleable solids to less than 0.1 mL/L.

According to Shannon & Wilson, Inc. (S&W 1985b) and Peterson et al. (1986), a number of miners discharge water containing less than 0.1 mL/L settleable solids by using settling ponds or by using settling ponds followed by filtration through tailings.

The Alaska water quality criterion for sediment, including settleable solids, is: "No measureable increase in concentrations of sediment above natural conditions" (ADEC 1985). Most placer mines exceeded this criterion in the past. Currently, some miners (10 to 20 percent) are able to discharge wastewater containing less than 0.1 mL/L settleable solids, thereby meeting the Alaska criterion.

Turbidity

Turbidity is an expression of the optical property that causes light to be scattered and absorbed rather than transmitted in straight lines through a water sample. Turbidity in water is caused by the presence of suspended matter such as clay, silt, finely divided organic and inorganic matter, plankton and other microscopic organisms. A nephelometer having a light source for illuminating the sample and a readout device indicating the intensity of light scattered at 90 degrees to the path of incident light is the accepted method for measuring turbidity. According to Peterson et al. (1985), turbidity prevents the growth and photosynthesis of green plants and can also cause fish to avoid otherwise suitable habitat and prevent them from seeing their prey.

Turbidity levels in natural clearwater streams are generally less than 5 to 10 Nephelometric Turbidity Units (NTU), but may be 25 to 50 NTU during periods of high water. Data from two reports (D&M 1976; R&M 1982) were compiled to assess turbidity levels at various locations throughout mining operations using settling ponds. Turbidity levels averaged approximately 9000 NTU in the sluice effluent and tailrace. Turbidity levels average about 7000 NTU in pond influents, 1700 in pond effluents and about 900 NTU after mixing in the receiving stream. EPA (1985) collected data at a number of mine sites over a two-year study period and reported the monthly average pond effluent turbidity level as 1400 NTU. Although operations having ponds release higher quality water than operations without ponds, turbidity levels at operations with ponds are usually about 100 times higher than natural conditions (R&M 1982). Settling ponds do not effectively remove the fine particles causing high turbidity levels. About 90 percent of turbidity is caused by particles less than about 10 microns in diameter and a well designed settling pond can only be expected to trap particles larger than about 10 to 20 microns. Hence, for many operations, a significant amount of dilution is necessary to meet the Alaska turbidity criterion. Therefore, placer mining operations normally cannot meet the turbidity criterion without a bypass or a large stream for mixing.

The Alaska water quality criteria for turbidity are: "Shall not exceed 5 NTU above natural conditions when the natural turbidity is 50 NTU or less, and not have more than 10% increase in turbidity when the natural condition is more than 50 NTU, not to exceed a maximum increase of 15

NTU" (ADEC 1985). With few exceptions, placer miners are unable to meet these criteria. At one site in 1985, a placer operation using 100 percent recycle, five settling ponds and gravel filters produced a final effluent containing only 3.5 NTU (Peterson et al. 1986). This was the only documented placer operation meeting the turbidity criteria in 1985. Currently, achievement of the Alaska receiving water turbidity criteria is primarily a function of the relative volumes of the effluent and the receiving stream. Results of recent studies (S&W 1985b; Peterson et al. 1986) indicate that some miners are able to reduce turbidity levels in the receiving stream to hundreds of NTU rather than thousands of NTU which was common only a few years ago.

METALS

Hardness

Hardness is the property of water attributable to the total alkaline earth content that can produce insoluble soaps. All divalent metallic cations cause hardness, but the principal ones are calcium and magnesium. Hardness normally ranges from approximately 16 to 116 mg/L as calcium carbonate in streams unaffected by placer mining (D&M 1986). These values indicate the water is soft to moderately hard. Hardness concentrations of 21 to 92 mg/L as calcium carbonate in mined streams (D&M 1986) are similar to the range in unmined streams. There is no Alaska water quality criterion for hardness. However, hardness concentrations are important because of the interaction between hardness and some toxic metals. Cadmium, copper, lead, nickel, silver and zinc are more toxic at lower hardness concentrations. Formation of metallic hydroxides and carbonates occurs at high hardness concentrations leading to precipitation of these metals. Calcium and other hardness causing elements may also create physiological protection such as on the gill surface of fish. However, the mechanism of physiological protection is not well understood.

Iron and Manganese

Iron concentrations are typically low to moderate in unmined streams, ranging from 0.01 to about 0.85 mg/L (D&M 1986). However, iron concentrations are quite variable in mined streams, ranging from less than the detection limit to over 500 mg/L indicating that placer mining significantly increases the iron concentration in some streams. The Alaska drinking water criterion for iron is 0.3 mg/L (ADEC, 1985) and the criterion for the protection of freshwater aquatic life is 1.0 mg/L (EPA, 1976). Consequently, both criteria for iron are exceeded in some placer mined streams on a site specific basis. Much of the iron released by mining in the ferrous state rapidly oxidizes to the ferric state and precipitates to the streambed giving it a rust-red to yellow color.

Manganese concentrations measured at 15 locations in unmined streams ranged from 0.006 to 0.051 mg/L and averaged 0.015 mg/L (Mack 1985). Ten measurements in mined streams ranged from 0.006 to 9.57 mg/L with an average of 2.46 mg/L. The Alaska drinking water criterion for manganese is 0.05 mg/L (ADEC 1982). This criterion was exceeded in 8

out of 10 mined streams measured. However, dilution and oxidation reduces manganese concentrations downstream.

Arsenic

In Alaska, arsenic exists naturally in many surface waters in concentrations of 0.01 mg/L or less (ADEC 1979). Placer mining may increase the arsenic content in streams by exposing arsenic-containing rocks to surface waters and by increasing the load of arsenic-rich sediments in streams (Wilson and Hawkins 1978). This can occur in areas where the primary lode gold mineralization is associated with the mineral arsenopyrite. Arsenic is released from the parent rock by mining as dissolved arsenite (As III). Arsenite oxidizes to arsenate (As V) under oxidizing conditions (Brown et al. 1982), which is the typical situation in Alaska streams. Arsenate can be removed from the water column by coprecipitation or adsorption onto hydrous iron oxides, aluminum hydroxide and clays (EPA 1980a). Adsorption of arsenate appears to remove arsenic from solution to the sediments and prevent high arsenic concentrations from being present in the water column (EPA 1980a).

Variability of total dissolved and total suspended arsenic concentrations in placer mined streams is quite high (R&M 1982). Arsenic is mainly transported with the suspended material in turbid streams. This is apparent from data presented by R&M (1982) where the mean suspended arsenic concentration below the sluice of 15 mines was approximately 200 times greater than dissolved arsenic. Removing the suspended and settleable solids from mining wastewater removes the majority of arsenic (Bainbridge 1979; R&M 1982; Kohlmann Ruggiero Engineers, PC [KRE], 1984; S&W 1985b). R&M (1982) found that total suspended arsenic decreased by 84 percent in a settling pond and data presented by Bainbridge (1979) display an 88 percent reduction in total arsenic in ponds. S&W (1985b) reports that a significant portion of arsenic settled out in pond systems where mean total arsenic and total dissolved arsenic concentrations were reduced by 67 percent for total arsenic and 95 percent for total dissolved arsenic. Although settling ponds are capable of removing arsenic that is bound up with particulate matter, dissolved arsenic and arsenic associated with fine suspended solids will not be removed by settling ponds alone. According to EPA (1985), arsenic is a component of TSS. Therefore, arsenic is excluded in the proposed EPA effluent criteria for the placer mining industry because this toxic metal is effectively controlled (or removed) by the technology of the effluent limitation for TSS.

The Alaska water quality standards (ADEC 1985) for toxic and other deleterious organic and inorganic substances are applied by using the Alaska Drinking Water Standards (ADEC 1982), EPA's Quality Criteria for Water (EPA 1976), or the ambient water quality criteria for 65 toxic pollutants including arsenic (EPA 1980b). The drinking water criterion for total arsenic is 0.05 mg/L and the criterion for the protection of freshwater aquatic life is that the concentration of total recoverable trivalent inorganic arsenic should not exceed 0.440 mg/L at any time (EPA 1980b). However, there is no currently acceptable method for measuring trivalent inorganic arsenic. Hence, the Alaska criterion for

total arsenic is 0.05 mg/L. This level is exceeded at some placer mines in Alaska.

Mercury

Natural mercury concentrations are so low that they are typically undetectable in Alaska streams. Concern regarding mercury at placer mines arises because of its use within the sluice box for amalgamation, a common practice in the old pick and shovel days of mining but a rare practice today.

Because of its high specific gravity, mercury is very susceptible to removal by settling ponds. Removal of the suspended and settleable solids by settling results in effective removal of mercury (Bainbridge 1979). However, mercury associated with small sediment particles or dissolved mercury will not be removed by ponds.

There are several reasons why miners do not use mercury today. Gold coated with an oxide will not unite with mercury. Hence, tests need to be conducted to determine whether gold in a particular placer deposit will amalgamate prior to using mercury. Another reason is economic. Mercury is not normally used in the upper part of the sluice box because it is economically beneficial to recover as much free gold as possible. Because free gold can be sold at the world market price, the miner will usually obtain 10 to 20 percent more for his gold if it is not treated with mercury. The use of mercury is also limited because of its high cost.

For the protection of freshwater aquatic life the criterion for total mercury is 0.0002 mg/L (EPA 1981). This criterion is applied by Alaska in receiving streams. Although mercury may be used by miners in rare instances, its loss has not been demonstrated to cause water quality problems. LaPerriere et al. (1985) indicate that mercury concentrations are not significantly elevated in mined streams.

According to EPA (1985), mercury is a component of TSS. Therefore, mercury is excluded in the proposed effluent criteria for the placer mining industry (EPA 1985) because this toxic metal is effectively controlled (or removed) by the technology of the effluent limitation for TSS.

Because of the relative absence of the use of mercury by placer miners and because its loss has not been demonstrated to cause water quality problems, mercury is not considered to be a critical parameter in placer mining in Alaska.

Other Trace Metals

Concentrations of many trace metals have been measured in mined as well as in unmined streams. Most of these metals are essentially unaffected by placer mining because their concentrations downstream from mining rarely exceed their respective criteria. For example, LaPerriere et al. (1985) note that cadmium concentrations are not significantly elevated in mined streams, but that copper, lead, and zinc

concentrations may be significantly higher in streams below active placer mining. This situation was also noted by D&M (1986) in the Crooked Creek drainage. Metal concentrations in unmined streams ranged from 0.0001 to 0.004 mg/L for lead and zinc and 0.001 to 0.035 mg/L for copper. Concentration ranges for these trace metals in mined streams were significantly higher: 0.022 to 9.57 mg/L for copper, 0.009 to 2.46 mg/L for lead and 0.098 to 0.882 mg/L for zinc.

The Alaska drinking water criteria for these metals is 1.0 mg/L for copper, 0.05 mg/L for lead and 5.0 mg/L for zinc. Copper and lead exceed their respective drinking water criteria in site specific instances. The Alaska receiving water criteria for the protection of freshwater aquatic life for these metals is: 0.012 mg/L for copper, 0.074 mg/L for lead and 0.180 mg/L for zinc when hardness is 50 mg/L (EPA 1980b). All three metals have exceeded their respective criterion for the protection of freshwater aquatic life at some placer operations.

TOXIC ORGANIC COMPOUNDS

Toxic organic compounds such as volatile and semi-volatile compounds, pesticides and herbicides are primarily synthetic and generally are not naturally associated with metal ores. EPA (1985) reported the results of analyses for the priority toxic organics for final effluent samples collected from 10 mines. Only two priority organics, methylene chloride and bis (2-ethylhexyl) phthalate, were detected in the final effluent at some of the mines. None of the remaining priority organics were detected at any of the mines. In the sampling for the priiority organics, 117 of the listed toxic organics were not detected and therefore were excluded from further consideration for effluent criteria. The two priority organics detected were also excluded because they were present in only trace amounts and not likely to cause toxic effects. These two organics were present in amounts too small to be effectively reduced by current technologies. The presence of these two priority organics in other mining industries has been attributed to sample and laboratory contamination and EPA felt such contamination was the source of these pollutants in placer mine wastewater as well.

SUMMARY

The adverse effect of placer mining on downstream water quality is mainly due to particulates and metals. All other parameters are unaffected by placer mining or are affected to only an insignificant degree. Most parameters in receiving waters downstream from placer mines meet the criteria of the Alaska water quality standards for all the protected uses. Turbidity and TSS are the most difficult parameters to control because of the problems encountered in removing small particles. Consequently, miners are unable to meet the Alaska turbidity and TSS criteria. Many placer operations discharge wastewater containing settleable solids exceeding 0.2 mL/L. Concentrations of arsenic, copper, iron, lead, manganese and zinc may exceed the Alaska criteria for drinking water, recreation or the protection of aquatic biota downstream from some placer mining operations on some occasions.

MONITORING WATER QUALITY PARAMETERS

One sample station should be located at the end of the mixing zone where mining wastewater enters the receiving stream to determine whether most parameters meet Alaska receiving water criteria. These criteria are set so as not to exceed a given value, to be less than a given value or are expressed as a range. The criteria for some parameters, such as settleable solids, turbidity, and TSS, are set in terms of not exceeding a certain level over background conditions. Hence, a control sample station is required upstream from the mining operation for these parameters. A third sample station should be at the mine effluent to monitor EPA National Pollutant Discharge Elimination System permit criteria. The proposed EPA criteria include 0.2 mL/L settleable solids and 2000 mg/L TSS (EPA 1985). Samples collected at all stations should consist of three or more grab samples across the stream. These samples should be combined to form a single composite. Also, parameters measured in the field such as dissolved oxygen and temperature should be measured at least at three points across the stream and averaged to obtain the reported value.

Although all the parameters previously discussed are important for characterizing water quality, some are more critical for monitoring the performance of placer mining. The most important parameters to include in a monitoring program are, in order of decreasing importance, settleable solids, turbidity, TSS, arsenic, iron, copper, lead, zinc and manganese. High levels of all these parameters adversely affect water used for drinking, recreation, or the protection of aquatic biota. Settleable solids is the most critical parameter because the solids cover streambeds smothering benthic organisms and spawning habitat. They may also pack the streambed with particles thereby reducing habitat diversity and affecting interstitial water flow. Turbid conditions reduce the number and diversity of organisms dependent on photosynthesis and adversely affect sight feeding organisms. TSS contain particles causing settleable solids and turbidity. Total arsenic, iron, copper, lead, zinc and manganese concentrations must be measured because levels of these metals have been shown to exceed criteria on a site specific basis. Total metals as opposed to total recoverable metals are measured because established criteria are in terms of the total fraction. Finally, hardness should be measured to relate the toxicity of copper, lead and zinc concentrations to criteria for the protection of aquatic biota.

Measurement of many water quality parameters should not be attempted by inexperienced people. Even sample collection should be coordinated by an experienced sampler to insure proper sample collection and preservation techniques are used. For example, specific types of containers (plastic or glass), bottle cleaning requirements, preservatives, and sample holding times are specified for each parameter. It is beyond the scope of this handbook to reprint the specific sample collection, preservation and analytical techniques for each parameter. Detailed descriptions of these techniques appear in Standard Methods for the Examination of Water and Wastewater (American Public Health Association 1985) and in EPA's Methods for Chemical Analysis of Water and Wastes (EPA 1983).

METHODS FOR MITIGATING WATER QUALITY DEGRADATION

Two techniques for improving downstream water quality, either separately or in combination, are employed in Alaska. One technique is reduction of water use. The second technique, includes various wastewater treatment technologies employed to improve effluent quality, such as settling ponds, effluent filtration and effective use of a stream bypass and the tailrace.

WATER USE REDUCTION

The volume of makeup water (fresh water added to the processing stream) required in an operation can be significantly reduced by either recycling, reducing the volume of water used in processing or employing a combination of the two methods. S&W (1985b) showed that recycling could reduce the volume of makeup water required to as little as 4 percent of the total process water required. Water use can also be significantly reduced by using paydirt classification, which is removal of oversize material from the processing stream. Complex factors affect water use in a sluice and the water volume used depends to a great extent on the type of material processed and the judgement of the mine operator. The volume of water used for sluicing varies from a few hundred to many thousand gallons per minute (gpm). The highest water use occurs in conventional sluice boxes with no classification, and averages about 3000 gpm. Water use in a sluice with undercurrents, a form of classification averages about 2500 gpm.

Water duty is the volume of paydirt moved per unit volume of water and is an appropriate measure of water use. S&W (1985b) defines water duty as the number of bank cubic yards of paydirt processed per 1000 gallons of water. Water duties at 24 mines (R&M 1982; S&W 1985b) ranged from 0.26 to 3.3 with an average of 0.88. Comparison of 23 sluice operations in these studies demonstrates that operations with feed classification use less water than operations without classification. The volume of water used per cubic yard processed by current mining practices can be computed from water duty. Using mean water duties from the studies cited above, the volume of water used to sluice one cubic yard of pay dirt is 830 gallons for operations with feed classification and 1820 gallons for no feed classification.

Currently, the most widespread and effective technique for reducing water use at Alaska placer mines is feed classification or classification within the sluice. A few operations use jigs, which significantly reduce water use. However, jigs are expensive and more complicated to operate.

Reducing water use by classification or other means results in improved downstream water quality as long as the operation is located off stream and a clearwater bypass is provided. Reduced water use results in better settling within the settling pond system and reduces effluent volumes. This situation along with a clearwater bypass results in lower sediment loading in the receiving stream.

TREATMENT TECHNOLOGIES

There are a number of methods to improve effluent quality. The most important of these methods is the settling pond. Other methods include effluent filtration and effective use of a stream bypass and the tailrace.

Settling ponds are the most common means for removing sediment from placer mining wastewater, and are a required treatment scheme in Alaska (Yost 1982). Of six mine operations studied by S&W (1985b), one used filtration through tailings and had no surface discharge. Total suspended solids removal at the other five mines ranged from 82 to 99.8 percent with an average removal of 92 percent. Turbidity removed in the ponds at these five mines averaged 70 percent and ranged from 13 to 86 percent. Because of the presence of clay and silt, the turbidity criterion is virtually impossible to attain with settling ponds alone. Settleable solids levels discharged from well designed and maintained settling ponds are quite low, 0.1 mL/L or less. Small ponds receiving substantial flow or ponds that have filled with solids normally discharge water containing relatively high settleable solids levels.

A bypass diverts that portion of the stream flow which is not needed for sluicing around the processing area and settling ponds. Improved settling pond performance and increased dilution of the final pond effluent are two water quality benefits of a bypass. Of the six mines studied by S&W (1985b), only one bypassed all excess flow around the settling ponds, and then only in the latter half of the summer. In this instance, water quality measured 500 feet downstream from the confluence of the mine effluent and receiving stream was better with the bypass despite the fact that the final pond effluent quality was worse. This is illustrated by mean levels of three parameters listed below.

	No Bypass		With Bypass	
	Final Pond Effluent	500 Feet Downstream	Final Pond Effluent	500 Feet Downstream
Settleable Solids, ml/L	0.59	0.72	4.0	0.48
Total Suspended Solids, mg/L	1625	1447	11,350	911
Turbidity, NTU	2020	2190	7300	1010

A tailrace is important because, depending on its gradient, substantial amounts of the coarse and fine tailing fractions can be removed. Flattening the gradient will reduce the solids volume loading on a settling pond but will increase the volume of coarse tailings material to be removed and stacked by mechanized equipment. A tailrace sufficiently long to permit the construction of a meandering path of flow will also allow the larger particles (sand size and larger) to settle out before reaching the pond (Sexton 1982; R&M 1983). Data presented by Bainbridge (1979) and R&M (1982) indicate improved solids removal in longer tailraces. There is no theorectical basis, however,

for a significant direct water quality benefit from a longer tailrace if the settling pond is adequately designed and maintained. However, premature filling of ponds resulting in decreased performance could be lessened if additional solids are removed in the tailrace.

Filtration may be conducted by directing wastewater through tailings dams or by direct discharge to tundra or vegetation. Efficiency of filtration through tailings depends on the particle size of the tailings, thickness of the tailings dam, physical characteristics of the suspended sediments and the volume of wastewater to be filtered. Filtration holds little promise as a primary treatment technique because the high concentrations of solids in an effluent will quickly pack a filter. However, filtration may be effective as a final polishing step downstream from settling ponds. Bainbridge (1979) and the Department of Indian and Northern Affairs (Yukon Territory 1981) each report field data showing that filtration may provide a relatively clear effluent or a very cloudy effluent. Of the six mines studied by S&W (1985b), only one employed filtration through tailings and this was in conjunction with a pond system. Water quality in the receiving stream was better than at the other mines studied. Settleable solids levels were always less than 0.1 ml/L, total suspended solids averaged 376 mg/L and ranged from 20 to 950 mg/L and turbidity averaged 450 NTU and ranged from 45 to 1100 NTU. Peterson et al. (1986) note that two 1985 field tests of filtration with tundra vegetation proved no more effective than settling ponds at removing turbidity. However, both tests were conducted at relatively high sediment loading rates.

WATER-RELATED REGULATORY REQUIREMENTS

The regulatory program for placer mining on National Park System lands in Alaska requires action by the Environmental Protection Agency (EPA), the Alaska Department of Environmental Conservation (State), the U.S. Army Corps of Engineers (Corps), and the National Park Service (NPS). Each agency has singular responsibilities to manage separate aspects related to maintaining the quality of water affected by mining.

At the heart of the regulatory program are water quality standards, which are established by the State. Standards are made up of the use or uses that can be made of water bodies and the water quality criteria necessary to protect that use or uses. The purposes of water quality standards are to protect public health and welfare, enhance the quality of the water, and to serve the broader purposes of the Clean Water Act. The standards serve as the regulatory basis for the establishment of water quality-based treatment controls and strategies beyond the technology-based levels of treatment required by sections 301(b) and 306 of the Clean Water Act.

Water quality criteria are elements of the water quality standards expressed as constituent concentrations that represent the quality of water that will protect a designated use. When the criteria are properly selected and met, it is assumed that the resulting water quality will protect the uses. The criteria are established through the use of bioassay or biological criteria, by adopting or modifying EPA's guidance or by using narrative descriptions where numerical values

cannot be established. The criteria adopted by the States in their water quality standards are enforceable requirements that are used to regulate man-made alterations and concern such constituents as dissolved oxygen, pH, fecal coliform bacteria, and temperature.

Alaska's water quality criteria, when used in combination with the water use designation, constitute the water quality standard for a particular water body. The water quality standards regulate man-made alterations to the waters of the state. The surface waters in National Park System units in Alaska are protected for all designated use categories. The most stringent criteria for each parameter among the various designated water uses is used to protect the receiving water quality. As part of the Clean Water Act's antidegradation policy, the state regulations prohibit reclassification (i.e., downgrading) of uses for waters within units of the National Park Service. The State has established criteria for turbidity and sediment, the two primary water quality criteria of concern for placer miners. The turbidity criterion requires that dischrges may not exceed five NTU above natural conditions when the natural turbidity is 50 NTU or less, and not have more than 10 percent increase in turbidity when the natural condition is more than 50 NTU, not to exceed a maximum increase of 25 NTU. the criterion for sediment requires that there be no increase in concentration of sediment, including settleable solids, above natural conditions.

Under the National Pollutant Discharge Elimination System (NPDES), permits are issued to discharges to ensure that discharges do not violate water quality standards following the general requirements of section 402 of the Clean Water Act. In the case of Alaska, the state does not have primacy for issuing NPDES permits. Such permits are issued by the EPA for all dischargers in Alaska. EPA issues the NPDES permits that contain water quality-based permit limitations which EPA believes meet the requirements of the state's receiving water quality standards. Before they go into effect, the state of Alaska must "certify" that the state's criteria will be met by the permit conditions prescribed under section 401 of the Clean Water Act. The state may enforce water quality standards directly, apart from the NPDES permit as well.

The EPA proposed effluent limitation guidelines and standards for placer mining in November, 1985. The draft guidelines define the technology-based discharge standards that must be met by placer mining operations. The guidelines, which define best practicable technology (BPT), best conventional technology (BCT), best available technology (BAT), and new source performance standards (NSPS), establish different requirements depending on the size of the proposed placer mining operation. According to EPA, these effluent guidelines meet the State's receiving water quality criteria.

The four classes of operations, based on their daily production rates, are: 1) fewer than 20 cubic yards; 2) all mines producing between 20 and 500 cubic yards; 3) all mines, except group 4 below, with production of greater than 500 cubic yards; 4) large dredges operating in self-contained ponds and producing more than 4000 cubic yards. EPA

concluded that effluent limitations for mines producing fewer than 20 cubic yards per day are not warranted at this time. As a consequence, EPA proposes that permit limitations will be developed based on the best professional judgement of the permit writer for the smallest class of operation. For the second class of operations, EPA proposes that the control technology for BPT, BCT, and BAT be simple settling ponds with a minimum six hour detention time and an effluent quality of 0.2 ml/l settleable solids and total suspended solids of 2000 mg/l. For the remaining two classes, EPA proposes that BCT, BPT, and BAT effluent limitation guidelines be based on total recycle of process water.

At this time, it is unclear when the final effluent limitation guidelines and standards will be promulgated. Until they are published as final , the NPDES permits issued will be developed on a case-by-case basis to meet the State standards.

In addition to NPDES permits, placer mining operations are required to obtain permits from the Corps under section 404 of the Clean Water Act. That section deals with the discharge of dredge or fill material into the waters of the United States. The Corps issues the section 404 permit, subject to certification by the State that State water quality standards will be met.

All mining operations in units of the National Park System are required to comply with the requirements of 36 CFR 9A which implement the Mining in th Parks Act. The regulations require, among other things, that operators prepare and submit plans of operation for NPS approval. Before the Regional Director of NPS may approve a plan of operations, he must assure that the applicant has taken all steps necessary to comply with any Federal, State, or local laws or regulations. For placer mining, at a mininmum, this would include both NPDES and section 404 permits. The applicant must also clearly set forth his proposed operating methods and outline his plans for protecting the resources of the park during mining. In addition, the plan requires the operator to describe his plans for reclaiming any lands disturbed in the course of his operations.

Each of these regulatory requirements is designed to ensure that placer mining operations do not adversely affect natural resources that are of national significance. Both the Mining in the Parks Act and the Clean Water Act were passed to ensure that activities permitted or licensed by the Federal government would not result in degradation of resources.

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CHAPTER 4

RECLAMATION

INTRODUCTION

This chapter describes reclamation procedures and practices applicable at typical Alaska placer operations. The term "reclamation" is used because the area may or may not be returned to its original use while "revegetation" only refers to the return of a vegetative cover and "rehabilitation" may refer solely to visual aspects (Bradshaw and Chadwick 1980). The objective of placer minesite reclamation is to return aquatic and terrestrial habitats to as near a premining state as possible. Habitats should be physically rehabilitated within a year following mining and biologically reclaimed in a timely manner. It should be noted that many placer mined areas cannot be reclaimed to their original condition because mining alters the soil conditions and thus the composition of vegetation that becomes reestablished. For example, the concentration of relatively coarse tailings and elimination of permafrost during mining results in warmer, well drained soils. Alterations in native soil conditions (e.g., texture, structure, compaction, drainage and organic soil content) may encourage the development of plant communities that did not exist previously. Furthermore, when ice-rich permafrost thaws, the local topography may be altered to the extent that it is infeasible to restore the original topography.

Alaska manifests a diverse environment spanning four climatic zones: arctic, continental, transitional and maritime. Climatic extremes range from cold, dry arctic conditions in the north and northwest, moderating somewhat through the Interior, to warmer, wetter conditions in Southeast Alaska. In general, climatic regimes relate closely to permafrost conditions: continuous, discontinuous, sporadic, and permafrost-free. Vegetation types are a result of climate, incident solar radiation, ground temperature, ground moisture, site history and soil type. The reestablishment of terrestrial plant communities is largely a function of climate, permafrost conditions and the depth of annual thawing (active layer). Successful reclamation of aquatic communities is contingent on site-specific stream hydraulic properties, ice conditions, substrate properties, incident solar radiation and water quality.

AQUATIC RECLAMATION

Aquatic reclamation involves the modification of stream characteristics to approximate natural conditions above and below a mine site. Stream reclamation should provide proper design flows, stream patterns and placement, hydraulic design and channel stability. Furthermore, stream reclamation can be refined to improve habitat quality to a condition even more suitable for aquatic organisms than the original stream conditions. Channel designs for fish-bearing streams should emphasize the development of habitat providing proper velocity, depth, substrate and cover.

Aquatic reclamation may require a significant amount of earthwork on the part of the miner. The most effective means of achieving the goal required is to develop a multi-year mining plan that recognizes reclamation objectives for placement of overburden stockpiles, settling

ponds, tailings piles and bypasses such that the areal extent of disturbed habitat at any one time is kept to a minimum.

STREAM CHANNEL

Gravelbed stream channels generally adjust their channel pattern, channel geometry and longitudinal profile through erosion and deposition. Factors affecting erosion and deposition processes include stream discharge, sediment transport, bed and bank material and valley slope (Hey 1982). Although infrequent flood events (e.g., 100-year floods) can transport large quantities of sediment, smaller floods having significant sediment transport capabilities have a greater effect on stream channels.

A major objective of aquatic reclamation is to design a stream channel which approximates pre-existing stream patterns, cross-sectional geometry, longitudinal profile and sediment-transport characteristics. Accommodations should be made for indigenous aquatic species in the reclaimed stream. A properly reclaimed stream requires minimal adjustment by the stream upon completion and should consequently reestablish natural equilibrium conditions rapidly.

Initial steps in stream reclamation include the assessment of acceptable flood risks in relation to the labor and materials required, selection of a flood recurrence interval for channel design and computation of corresponding discharge. The flood recurrence interval denotes the probability that a flow of a particular magnitude will occur within a stated period of time. Once acceptable recurrence intervals have been selected, corresponding stream discharges must be evaluated. High and mean annual stream flows of varying recurrence intervals may be predicted for Alaska streams on the basis of drainage area and mean annual precipitation, by using equations developed by Parks and Madison (1985). Alternative equations developed by Ashton and Carlson (1984) for high spring flows in southcentral and interior Alaska may also be used. Once a design-flood recurrence interval and discharge are computed, an appropriate stream pattern must be selected.

Stream Pattern

The stream pattern of the reclaimed stream channel should approximate the pattern which existed prior to mining. Channel patterns used in stream reclamation include meandering, mountain and braided.

In selecting an appropriate stream pattern, actual field observations and data should be used at sites which have not been disturbed by upstream mining. Data which should be collected and mapped to scale include stream course, number and length of pools and riffles and the location and number of channels within a given reach of stream. These data should be collected from actual field surveys whenever possible, but may also be determined from aerial photographs or topographic maps. The length, number and degree of bends and the number and location of pools and riffles in reclaimed streams should be similar to the natural stream pattern.

In valleys which have been mined to the extent that natural stream patterns cannot be determined, theoretical data must be used to evaluate an appropriate stream pattern. Leopold and Wolman (1957) indicate that the categorization of streams into braided, meandering, and straight channel patterns may be difficult. Several studies (Leopold and Wolman 1957; Schumm and Khan 1972; Bray 1982) indicate that the causative factors for meandering and braided channel patterns cannot be clearly differentiated. However, trends in stream pattern are related to channel and valley slopes, bankfull discharge and in some instances, bed-material size. For example, if the slope of the valley is gentle enough, a meandering stream may be considered. However, if the slope is too steep for a meandering stream, then a mountain stream pattern should be considered. If the bedload transport in the stream is expected to be high during summer flow levels, a braided pattern may be considered.

Meandering Streams - Considerations in the design of a meandering stream channel include channel geometry and bed stability, channel pattern, riffle and pool-spacing, location, cross section and profile. An approach for evaluating the general hydraulic geometry of meandering streams is presented by Rundquist et al. (1986).

The channel pattern of a meandering stream is determined by its sinousity and meander arc length (Hey 1983). The sinuosity is the ratio of the channel length to the valley axis length. In general, the sinuosity of meandering streams should be between 1.3 and 2.5. The meander arc length is the channel distance between two points of inflection. The number of meander arcs at a site can be calculated by knowing bankfull channel width, sinuosity and the length of the site. The number of meander sequences is simply half the number of meander arcs over a specified distance.

Placement of pools and riffles is critical in the design of meandering streams. The depth, length and spacing of pools depend on the radius of the meander arc, sediment load and stream size (Karaki et al. 1974). Generally, pools and riffles occur alternately every 5 to 7 channel widths apart (Leopold et al. 1964). At low flows, the water surface slope is gentle in pool areas and steep in riffle areas. As flows increase, the water surface slope in pools increases while the riffle water surface decreases its steepness.

Mountain Streams - Stability is the major design consideration in reclamation of mountain stream channels. High channel velocities erode small materials on the stream bed and banks. Therefore, large rocks should be used to armor the channel.

Simons, Li and Associates (1982) used previously established relationships between stream velocity and channel roughness to design channel geometry, discharge and depth of flow in mountain streams. The optimum shape for a mountain stream is dependent upon site-specific parameters, including the 2-year high flow, the median riprap size available and the channel slope.

In natural mountain streams, as in meandering streams, the riffle-pool sequence repeats about every 5 to 7 channel widths. By alternating flatter sections and shorter, steeper sections along the length of the channel, a natural riffle and pool sequence can be created.

The reclamation of mountain streams for aquatic habitat should focus on the design of channel structures and modifications that create habitat diversity. The creation of uniform, straight-channel streams that have little or no structural diversity should be avoided.

Channel modifications such as the placement of boulders, rock islands or submerged-rock weirs can be used to enhance habitat diversity for aquatic invertebrates and provide rearing habitat for fish in mountain streams. The placement of boulders and rock weirs must be adequate to allow fish passage at both high and low flows.

Braided Streams - Since, by nature, braided channels are unstable and adjust their position frequently within the stream valley, their configuration is less critical than for meandering or mountain streams. Design parameters for braided streams include channel geometry, channel pattern and channel profile.

Braided streams are characterized by unstable beds, in which the transportation of bed material occurs at relatively low flows. Therefore, any attempt to increase bed stability in braided streams is unnecessary. Braided streams should be constructed with a main channel that carries most of the flow and several high-water channels that carry some flow.

Stream Cross-Section

The cross-section of a reclaimed stream channel should approximate the condition which existed prior to mining, or alternatively, the condition which occurs above or below the mine site. If data are unavailable, the stream cross-section should be based on theoretical design-flow using an appropriate recurrence interval. Usually, the accepted flood event for defining the channel cross-section is the bankfull flood (Leopold and Wolman 1957; Nixon 1959; Parker 1979; Hey 1982).

Stream Length

Stream length design is a critical element of stream reclamation. If the stream length is shorter than its original length, the slope will be steeper causing erosion at the upper end of the site and deposition in downstream reaches. In contrast, if the stream is longer than the original length, material will deposit in the upper end of the site and will erode in the downstream reaches.

Stream Placement

If the valley is mined on one side first, then on the other, final stream placement should be made along the valley wall that was mined first. This minimizes the number of temporary diversions required

during mining and provides for final stream placement at the earliest possible time. Within the constraints imposed by the stream pattern selected, the stream may be placed so that it shifts from one side of the valley to the other. The stream channel should be placed to avoid erosion of settling ponds and overburden stockpiles.

Streambed Elevation

Elevation of the reclaimed stream bed should be similar to that of the original stream bed. If the stream is located lower than its natural level, erosion may occur at the upstream end. If the channel were placed too high, deposition may occur at the upper end, while erosion will occur further downstream. New erosion and deposition resulting from incorrect stream elevation (or length) are natural processes through which the stream adjusts its pattern to equilibrium conditions.

Channel Stability

Channel stability is the capacity of a stream to resist erosion of bed and bank materials and the lateral movement or possible failure of the channel. Bed stability of a reclaimed stream may be evaluated by comparing the slope of the reclaimed stream to the equilibrium slope. The equilibrium slope is the slope at which bed material is about to move for the given flow, geometry and bed material size. According to Simons, Li and Associates (1982), the channel is assumed to be stable if the channel slope is approximately equal to or less than the computed equilibrium slope. If the channel slope is greater than the equilibruim slope, the channel bed is probably unstable. Where greater bed stabilities are necessary, an armor layer (boulders) can be constructed or the size of existing armor layer materials can be increased.

Several layers of different sized armor material should be used in constructing a bed armor layer. A gravel layer at least 6 inches deep should be laid down first (Joyce et al. 1980). In low velocity streams, a subsequent layer consisting of large cobbles and small boulders should be placed above gravel-sized material. In higher velocity streams, riprap (large rocks or boulders) may be required in areas where bed scour should be avoided. The shape, size and type of rock are the important considerations in selecting erosion-resistant riprap materials. The rock should be hard, dense and durable. A gravel filter may be necessary behind the riprap layer to prevent the loss of bank material through voids in the riprap.

Erosion of channel banks and lateral movement of the channel may be avoided by protecting the bank with revetment. However, revetment will preclude vegetative development on the banks.

AQUATIC HABITAT

Placer mining can reduce stream habitat quality for periphyton, aquatic invertebrates and fish populations. Reclamation plans should include an assessment of populations occurring in or near the mined stream, an evaluation of the seasonal habitat utilization patterns of fish and

ideally an attempt at restoring both population and diversity to premining levels. Reclamation design should include features that allow use of the reclaimed site by indigenous aquatic species. In some instances, stream channel reclamation may be used to create habitat conditions which are more suitable to fish than natural conditions. The selection and placement of features for the restoration of aquatic habitat in mined streams depends on the stream type, the value of the stream as aquatic habitat prior to mining and the degree of habitat alteration caused by mining.

Habitat diversity is one of the key elements to successful stream reclamation. Channel structure not only influences streamflow components of the aquatic habitat, but may also influence biological factors such as aquatic food production and habitat utilization by macroinvertebrates. Channel design may be used to increase the diversity of depths and velocities in the reclaimed stream. By designing for streamflow diversity, suitable depth and velocity conditions are provided for a variety of organisms over a range of flow levels. Habitat diversity may further be enhanced by creating heterogeneous substrates composed of different size bed materials. Within the constraints imposed by other environmental parameters in the system, substrate diversity facilitates the development of diverse benthic macroinvertebrate communities. Other elements of habitat reclamation include the addition of stream stabilization structures to enhance pool areas and the presence of cover for fish. Streamside vegetation is also important for modifying aquatic habitats, stabilizing streambanks and as a nutrient source for the stream.

Reclamation should be concerned with fish passage in streams supporting important fisheries. Passage requirements are primarily related to depth and velocity criteria. Depths and velocities must be within an acceptable range so that fish can utilize available habitat at and above the mine site. Consequently, passage requirements should be included in the channel-design phase of stream reclamation. Once the stream channel has been designed for fish passage, modifications can be made to the stream to enhance rearing habitat.

The reclamation of rearing habitat usually requires a higher degree of complexity in channel design than simply providing for passage. There are a number of methods for improving rearing habitat during stream reclamation. Submerged logs can be lashed together and held in place by rocks or vegetation. Another method is to anchor trees, large branches or root bundles to the stream bank. Rearing habitat can also be created by placing large rocks or boulders directly in the stream. Wing deflectors (rocks placed along one bank) are considered to be one of the most useful devices for improving fish habitat (White and Brynildson 1967). These structures guide and concentrate stream flow, thereby increasing the depth of pools and the angle of slopes on banks and bends. In some situations, submerged rock weirs may be preferable to wing deflectors. The restoration of spawning habitat, particularly for salmon, may involve considerable biological and hydraulic engineering design.

SETTLING PONDS

The approach taken for settling pond reclamation depends on the location of the settling pond relative to the final reclaimed stream channel and the amount of organic and fine inorganic material available at the site. The preferred alternative is to site the pond to avoid the active floodplain of the reestablished stream. In instances where settling ponds are situated within the active floodplain, it may be necessary to remove fine sediments from the settling pond. However, sediment removal is difficult and creates problems of stabilizing the sediment at a new location. Any sediments removed from the ponds should be placed where they will not drain directly into the stream and should be located in areas where they are protected from the erosive action of stream discharge and surface drainage. The construction of berms between sediment stockpiles and the stream will prevent silt from being washed into the stream.

Some of the fine sediments removed from the pond may be spread over graded tailing piles when there is an insufficient amount of suitable inorganic soil material at the site. Alternatively, the sediments may be stabilized in place by placing a protective armor of oversize material over the pond. When the settling pond is reclaimed in place, lateral migration of the stream channel into the reclaimed settling pond may be prevented by using riprap where necessary.

TERRESTRIAL RECLAMATION

The primary goal of terrestrial reclamation is to return the area of disturbance to as close to a premining condition as possible. This process involves the stabilization of floodplain materials, reestablishment of natural vegetative communities and the promotion of vegetative recovery (Dick 1974). Rehabilitated mine sites can be expected to support different wildlife species at different stages of vegetative succession (Durst 1984). Reclamation of terrestrial habitat encompasses site grading throughout the entire area of disturbance, dispersement of organic overburden stockpiles and revegetation of riparian habitat, inactive floodplains and valley terraces (Keller and Leroy 1975).

Site grading involves the reduction of tailings piles to specified slopes for control of site drainage patterns, the stabilization of settling ponds and the control of site erosion and sedimentation. The first task of terrestrial reclamation is to modify the topography by grading the tailings piles. This serves to minimize erosion (Becker and Mills 1972), reduce the potential for nonpoint source pollution (Leroy 1973) and to enhance revegetation (Becker and Mills 1972; Leroy 1973).Proper contouring provides for high water discharge and sediment catchment basins. Final slope grading and contouring are especially important in riparian habitats where it is critical for the stream to remain in its designed channel for the first several years. Following final site grading and contouring, stockpiled overburden and vegetative slash can be dispersed to enhance the recovery of native vegetation. Revegetation can be accomplished by broadcasting vegetative debris, planting viable portions of live shrubs, transplanting live vegetation,

and seeding (Saville 1972; Billings 1974; Dabbs et al. 1974; Johnson and Van Cleve 1976; Joyce 1980; Johnson 1981). In addition, woody vegetation, tree trunks and rock piles can be strategically located to attract small mammals and birds.

Terrestrial reclamation may require a significant amount of earthwork on the part of the miner. As with aquatic reclamation, the most effective means of minimizing the work required is to develop a multi-year plan that recognizes the reclamation objectives and establishes erosion control during mining by placing overburden stockpiles, settling ponds, tailings piles, ditches, and bypasses accordingly.

OVERBURDEN DISPERSEMENT

The replacement of organic overburden should be carried out in a manner which assists in returning the site to as close to premining conditions as possible. At sites where sufficient quantities of organic overburden are not available, inorganic fines from settling ponds may be used in lieu of organic material.

One important consideration regarding overburden is whether more harm will be done by redisturbing the overburden stockpile. Miners typically move overburden upslope away from the area to be mined. Vegetation is quickly reestablished in the stockpiled-overburden area, sometimes within less then 1 year. Hence, the choice becomes one of moving overburden containing established vegetation in order to cover tailings piles, thereby creating a potential for more erosion, or leaving the overburden in place recognizing that it will take longer for vegetation to become established on the contoured tailings. Without any overburden, some tailings piles may remain unvegetated for more than 30 years (Holmes 1981). A useful compromise is to use a frontend loader to move some overburden and vegetation to selected areas in the tailings to establish "islands" of vegetation.

Although the rate and extent of natural revegetation varies dramatically with prevailing climatic conditions, the greatest impediment to natural revegetation is often the lack of moisture in the substrate (Berry 1970, Johnson et al. 1981). Natural revegetation is accelerated in areas where moisture can be retained in the organic material (Berry 1970). Before overburden stockpiles are distributed, a layer consisting of fine sediments, sand and fine-textured organic soil should be applied to retain moisture. The minimum thickness of this layer should be at least 10 cm. Additional benefits may occur with up to 50 cm or more of overburden depending on subsoil and climatic conditions.

REVEGETATION

Revegetation serves to enhance the recovery of native plant species which in turn reduces nonpoint source pollution and creates wildlife habitat (Dick 1974). The most effective techniques for enhancing natural revegetation include broadcasting shrub debris and transplanting intact tundra plots (Dick 1974; Joyce 1980).

Surface broadcasting entails the distribution of viable stems, branches and roots of small shrubs across the upper surface of organic soil layers (Joyce 1980). When woody plant parts are broadcast on soils having adequate moisture retention, the revegetation process is accelerated through the immediate development of mid-successional stage species which are shrubs (Viereck 1970). These materials should be distributed as soon as possible and at least within one year of stockpiling if at all possible (Rundquist et al. 1986).

Large-scale transplanting can also be an effective technique for enhancing revegetation. The most successful method for transplanting is to collect large tundra plots using a front end loader, as the material is being stripped for future mining. Although there is no optimal plot size, the survival rate of vegetative material is higher when large plots are removed. In general, the plot should contain 12 to 18 inches of soil containing undisturbed surface vegetation (Zasada and Epps 1976). Considerations for plot storage include protecting the material from surface runoff and high streamflow and preventing dessication.

At some sites it may be necessary to stabilize steep slopes by seeding with annual grasses, but only when they contain exposed erodible materials. The proper methods for seeding an area are described in Miller et al. (1983). Ideally, native grass seed should be raked into surface soil layers by dragging a section of chainlink fence behind a vehicle. If necessary, the area can be lightly fertilized to speed the revegetation process. Exotic species must be avoided, because they may impede reestablishment of native plants (Johnson 1984).

Any attempt toward revegetation should occur during the late spring to early summer or at freeze-up. If seeding is necessary, it should be completed by mid-summer or at freeze-up if site work does not allow seeding until late summer.

RIPARIAN HABITAT

The riparian habitat is the area adjacent to the active stream channel. This area supports terrestrial vegetation adapted to specific soil moisture and nutrient conditions near the stream. If the stream either supports fish or is being rehabilitated to provide fish habitat, overburden material should be distributed in the riparian habitat to enhance revegetation. The features developed in the riparian floodplain are dictated by hydrologic and engineering constraints imposed by stream channel design.

For meandering streams, the bankfull elevation (top of the stream bank) may be used as a reference for grading the active floodplain elevation. Although a grade of 50 to 1 is recommended on the inside of meanders (ADEC 1979), site constraints may dictate the use of a slope with maximum steepness of 10 to 1. On the outside of the meander, floodplain slopes should be graded to a maximum steepness of 3 to 1. Overflow channels may be required between some meander bends to carry water greater than bankfull. A small berm should be constructed at the

downstream end of such channels. A small terrace should be constructed on the outside of meander bends to minimize runoff and promote the development of riparian vegetation.

In mountain streams, terraces can be constructed on either side of the channel to collect sediments and channel high-water flows. The slope of such terraces should range between 10 to 1 and 50 to 1 away from the bankfull stream channel (ADEC, 1979).

Braided streams should be designed with a main channel to contain the bankfull flow along with highwater channels for carrying higher flows. The channels should wind through the floodplain in a braided-stream fashion. In braided floodplains, riparian habitats can be created from graded tailings between active channels. The upstream end of these "islands" should be stabilized by placing riprap to minimize erosion. Tailings piles within the active floodplain of a braided channel should be graded parallel to the direction of flow and stabilized to conform to the stream flow pattern.

After the slopes and elevations of the riparian habitat have been established, organic material should be placed in areas that provide the greatest benefit for revegetation. Although a single flood event could potentially erode this material before vegetation becomes established, with proper hydraulic design the potential benefits to terrestrial rehabilitation probably outweigh the risks. In addition, protective devices can be installed at all locations where organic material is to be placed. Materials may include tree stumps, tree trunks and large rocks. These devices will reduce the amount of material lost if a moderate flow event occurs.

UPLAND HABITAT

The inactive floodplain is the area between the riparian habitat and the 100-year floodplain. Settling ponds and tailings piles are often situated on portions of this upland habitat. As with the riparian habitat, the final topography of the inactive floodplain is dictated by the physical configuration of the site and the stream channel location and pattern. The inactive floodplain should be graded to allow flood water to spread out as shallow flow on the floodplain while still containing the flow. The final topography of the inactive floodplain should include slope grading for revegetation and wildlife habitat (Stoecker 1982) to as closely approximate premining conditions as possible.

Final grading of the inactive floodplain should allow for the dispersement of organic material to enhance the establishment of shrub thickets. These thickets provide buffer strips for wildlife migration between undisturbed vegetation upland and the area being mined. They may also enhance aquatic habitat, as previously discussed.

Sediment catchment basins can be excavated where necessary on the inactive floodplain to collect surface runoff and to retain water during flood events. These basins serve to reduce nonpoint source

pollution and enhance revegetation. Such basins should be designed to conform to the valley topography and should be large enough to intercept overland flow.

Settling ponds can be modified by developing diverse shoreline configurations, islands and variable water depths to provide habitat diversity for birds and fish. If necessary, accumulated sediments can be removed from the settling ponds and used to enhance revegetation. These sediments should be placed on the side of the pond opposite the active channel. In some instances, a berm may be required to contain the sediments and prevent them from returning to the pond. The bottom of the pond can be modified to create deep water areas near the center and shallow littoral zones near the shore (Joyce et al., 1980).

The valley terrace is a relatively level upland habitat situated outside of the 100-year floodplain. The goals of final grading on the valley terrace are to reestablish stable topography, enhance natural revegetation and create wildlife habitat. Specific features include diverse pond habitats, catchment basins and wildlife migration corridors.

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Glossary

Alevin - Newly hatched fish when still attached to the yolk sac.

Allochthonus - Material which arises from an area other than the one in which it is found, e.g., leaf detritus from a forest which is ultimately deposited in a stream or lake.

Alluvial - Transported and deposited by flowing water. Related to sediment deposits produced by flowing water.

Amalgamation - Process used to recover placer gold in which mercury is combined to form an alloy.

Anadromous - Migrating up rivers from the sea to breed in fresh water.

Armor layer - A layer of sediment that is coarse relative to the material underlying it and is erosion resistant to frequently occurring floods.

Auriferous - Gold-bearing material.

Autochthonus - Material which arises in an area in which it is found, e.g., photosynthetic production of algae in lakes or streams.

Bank measure - Gravel measured in place prior to its excavation and subsequent swelling.

Barren gravel - Gravel deposit that does not contain placer gold.

Beach placer - Deposits reconcentrated from the coastal plain gravels by waves and longshore currents.

Bedload - Sediments rolled and bounced along the bottom of a stream or sluice by moving water.

Bedrock - Solid rock underlying placer gravels.

Bench - A relatively flat area occurring on a valley wall.

Bench placer - Gravel deposit remnants of ancient alluvial fill that are left stranded along the sides of the valley above the present stream.

Benthic - Referring to the bottom of a body of water or to the organisms living in association with the bottom. These organisms can be attached, creeping, or burrowing forms.

Booming - A variation of ground sluicing in which water is stored in a reservoir and suddenly released to provide a large volume rush of water to erode and transport the paydirt.

Boreal - Of or pertaining to the forest areas and tundras of the North Temperate Zone and Arctic regions.

Braided stream - A river containing two or more interconnecting channels separated by unvegetated gravel bars, sparsely vegetated islands, and, occasionally, heavily vegetated islands. Its floodplain is typically wide and sparsely vegetated, and contains numerous high-water channels. The lateral stability of these systems is quite low within the boundaries of the active floodplain.

Bucketline dredge - A dredge in which the material is excavated and lifted to a floating platform and recovery plant by an endless chain of buckets.

Churn drill - A drill which successively raises and drops a heavy string of tools suspended from a drill line. Successive blows chop up the gravels and deepen the hole.

Classification - Seperation of particles in a fluid or in air by size.

Classifier - Equipment used to classify or seperate placer alluvium into different sized fractions with the use of screens.

Cleanup - Collecting concentrated placer material from the sluice or other recovery devices and furthur concentrating to seperate the gold particles.

Color - A relative term used to describe fine gold particles.

Concentration - Mechanical removal of lighter and valueless material from placer material to produce a gold-bearing concentrate. In placer mining the difference in specific gravity between gold and other sediment particles is used for seperation in a water media.

Depositional - The settling of material from the water column to the stream bed.

Detritus - Material released by weathering processes and subsequently transported and deposited as sediments by erosional forces.

Discharge - The rate of flow, or volume of water flowing in a given stream at a given place and within a given period of time.

Dredge - A floating machine which is usually mounted on a flat-bottomed hull and provided with the necessary equipment to dig, process and dispose of placer gravels in large river, gravel plain, beach and offshore deposits.

Eluvial - Loose material resulting from decomposition of rock in place.

False bedrock - A hard or tight formation within a placer deposit above the true bedrock.

Fine gold - Small gold particles. There is no standard definition but 20 to 100 mesh is used here.

Floodplain - The relatively level land composed of primarily unconsolidated river deposits that is located adjacent to a river and is subject to flooding; it contains an active floodplain and sometimes contains an inactive floodplain or terrace(s), or both.

Grizzly - Iron grating or heavy duty screen used to prevent large rocks from entering a sluice or other recovery equipment.

Habitat - The place where a population of organisms live and their surroundings, both living and nonliving; includes the provision of life requirements such as food and shelter.

Hardness - The property of water attributable to the total alkaline earth content that can produce insoluble soaps. All divalent metallic cations cause hardness, but the principal ones are calcium and magnesium.

Herbaceous - non-woody plants.

Hydraulicking - A soil loosening operation which uses water at high pressures to move earth downgrade. It is commonly used to thaw frozen material.

Jig - A gravity concentrating device in which gold is seperated from lighter minerals by pulsation and suction of water through a screen.

Lode - A vein or tabular mineral deposit in the country rock.

Macrophytes - Emergent or submerged vascular plants growing along stream and lake margins, in shallow pools, and in riffles.

Manifold - A pipe used to discharge water through a number of small openings.

Meander Wave Length - The average downvalley distance of two meanders.

Meandering Stream - A stream winding back and forth within the floodplain. The meandering channel shifts downvalley by a regular pattern of erosion and deposition. Few islands are found in this type of river and gravel deposits typically are found on the point bars at the insides of meanders.

Mountain Stream - A single channel, high gradient stream.

Muck - Usually frozen overburden overlying placer gravels in Interior Alaska composed of fine silt and organic matter.

Nephelometer - An instrument used for measuring turbidity consisting of a light source for illuminating the sample and a readout device indicating the intensity of light scattered at 90 degrees to the path of incident light.

Nephelometric Turbidity Unit (NTU) - A measure of turbidity determined by the amount of light scattered by solid particles in a water media.

Nonpoint-Source Pollution - All turbidity, suspended sediment, and sedimentation resulting from soil erosion caused by human activity and emanating from a widespread area.

Opencut - Placer mining process in which paydirt is excavated at the surface.

Overburden - Any organic or inorganic, worthless or low-value material lying on top of placer gravels.

Panning - Washing placer gravel by hand in a gold pan to concentrate and recover gold.

Paydirt - The gravels in an alluvial deposit which contain sufficient placer gold to mine at a profit.

Permafrost - Any soil, rock, or mixture thereof which remains below 32 degrees F for two years or more. Continuous permafrost occurs where all subsurface materials below the active layer (the surface layer which thaws annually) are frozen. The only interruption by thawed zones occurs under rivers or large lakes. Discontinuous permafrost displays a dominance of frozen material which is occasionally interrupted by thaw zones. Sporadic permafrost is dominated by thawed materials which is occasionally interrupted by frozen zones.

Pinched sluice - A device used for gravity concentration consisting of a wedge-shaped trough which tapers to a narrow opening at the discharge end. Gold migrates to the bottom of the trough at the discharge end and lighter minerals are forced upwards forming a stratified slurry. Gold is then removed by a splitter.

pH - The hydrogen-ion activity of a solution, which is an indication of the solutions acidity or basicity. A pH of 7.0 is neutral.

Placer - Deposits of unconsolidated sand, gravel and other detrital or residual material containing a relative concentration of gold or other valuable minerals. The valuable minerals have accumulated through natural processes of weathering, erosion, sedimentation and mechanical concentration.

Point-Source Pollution - Pollution which originates at a single entry point into a body of water.

Punch plate - A plate with holes drilled or punched through it to form a screen.

Resident Fish - Fish that remain in fresh water for their entire life cycle.

Riffles - A shallow rapids in an open stream, where the water surface is broken into waves by obstructions wholly or partly submerged.

Riparian - Pertaining to the banks of a stream or other body of water.

Riprap - Large sediments or angular rock used as an artificial armor layer.

Rocker - A short sluice-like trough fitted with transverse curved supports, permitting it to be rocked from side to side, and also fitted with a shallow feed hopper at its upper end.

Salmonid - Resembling or characteristic of a salmon, which includes the salmon, trout, and whitefishes belonging to the family Salmonidae.

Salting - Intentional, fraudulent salting where gold particles are placed in a sample to make it appear richer. Or, unintentional enrichment of a sample by erroneous or careless sampling procedures.

Settleable Solids - The volume of matter in water that will settle in one hour under quiescent conditions in an Imhoff cone.

Settling Ponds - Artificial structures designed to remove sediment from waters draining a mining operation.

Sinuosity - A measure of the amount of winding of a river within its floodplain; expressed as a ratio of the river channel length to the corresponding valley length.

Sluice box - An elongated trough, fitted with riffles, through which gravels are washed to recover gold or other heavy minerals.

Slurry - Mixture of solids and liquids capable of viscous flow.

Specific Gravity - A substance's weight compared with an equal volume of water. Gold is approximately 19 times heavier than water and has a specific gravity of about 19 depending on its purity.

Spiral concentrator - A wet type concentrator in which a sand and water mixture flow down a long spiral launder and separate into concentrate and tailings fractions. Concentrates are taken off through ports while the tailings flow to waste at the bottom.

Sponge - Porous mass of gold remaining after mercury has been removed from the amalgam by heating or solution in nitric acid.

Substrate - Stream bottom materials including silts, sands, gravels, cobbles, boulders, and bedrock.

Suspended Sediment - The portion of stream load moving in suspension and made up of particles having such density of grain size as to permit movement far above and for a long distance out of contact with the stream bed. The particles are held in suspension by the upward components of turbulent currents or by colloidal suspension.

Suspended Solids - The portion of organic or inorganic material in water that is retained on a glass fiber filter.

Tailings - Material processed through a placer operation usually consisting of coarse sand and larger particles.

Terrace - An elevated abandoned floodplain formed as a result of stream degradation and that is expected to be inundated only by infrequent flood events.

Trommel - A heavy duty revolving screen used for washing gravel and removing oversized rocks from the placer material before treatment by a sluice or other recovery equipment.

Turbidity - An expression of the optical property that causes light to be scattered and absorbed rather than transmitted in straight lines through a water sample. Turbidity in water is caused by the presence of suspended matter such as clay, silt, finely divided organic and inorganic matter, plankton, and other microscopic organisms.

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